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Significance of Mineralogy in the Development of Flowsheets for Processing Uranium Ores

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TECHNICAL REPORTS SERIES No.196

SIGNIFICANCE OF MINERALOGY IN THE DEVELOPMENT OF FLOWSHEETS FOR PROCESSING URANIUM ORES

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FOREWORD

This report has been prepared from material developed at and subsequent to a consultants' meeting held in Vienna in January 1978. The main purpose of the meeting was to prepare a document in the form of a guide for planning and developing treatment flowsheets for uranium ore processing.

It was apparent that ore mineralogy, analysed, described and interpreted in ways most meaningful to the metallurgist, is the most essential information required for forming the basis of such planning. This topic, here termed metallurgical mineralogy, is therefore a major theme of this publication.

In preparing the report the Agency has borne in mind the important need to impart the experience and knowledge gained in the more developed countries to those who are in the early stages of exploiting their uranium resources.

The contents may be criticized as lacking, in some respects, the requisite depth and detail of treatment. The Agency and the consultants are conscious of the need to expand the information in a number of ways. However, the report is presented in its present form in the belief that, as the first attempt to correlate, on a world-wide basis, ore type with processing, it will be considered as a useful basis for future development of these themes.

The IAEA wishes to express its thanks to the consultants who took part in the formulation of this report. Particular gratitude is due to the chairman, Mr. H.E. James, of the Atomic Energy Board, South Africa, who made a notable contribution to the work. The final compilation was the responsibility of Mr. F.R. Hartley of the Agency's Division of Nuclear Power and Reactors.

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PART I

PRINCIPLES AND PRACTICE OF URANIUM MINERALOGY AND ORE-PROCESSING METALLURGY

INTRODUCTION

The amount of uranium which can be mined and recovered as a marketable product, and the costs of mining and processing, are the two key factors that determine whether a uranium deposit can be exploited economically.

The mineralogy of a uranium ore fundamentally influences the technology and economics of its exploitation and determines the ultimate quantitative uranium recovery that can be achieved by any particular process. Therefore, in planning a programme of investigation into the extraction of uranium from any ore, a knowledge of its mineralogy and the behaviour of the ores in particular process environments is important.

Although it is not infrequent that the mineralogist is brought in to explain treatment difficulties that have been encountered by the extractive metallurgist during the course of an investigation, the adequate application of mineralogical data is seldom practised in the initial planning of an investigation into a treatment process. The metallurgist experienced in the technology of uranium ore processing will use his knowledge of mineralogy subconsciously in defining a treatment problem. Indeed, the nature of his experience is an intuitive understanding of the type of treatment flowsheet most likely to pertain to the particular ore mineralogy. Perhaps for this reason, the significance of mineralogy in the development of flowsheets for the processing of uranium ores is often understated.

Rocks are complex assemblages of minerals and differ widely in detail of composition and texture so that no single ore is exactly similar to another. Samples from one orebody vary, often significantly. These differences mean that at no time can a flowsheet be realistically optimized for all likely variations, nor can the treatment flowsheet be transferred in detail from one ore to another. A flowsheet is always a compromise of many factors and its development remains dynamic throughout the life of the mine. A knowledge of the mineralogy of the ore, the variations of that mineralogy and an appreciation of the behaviour of minerals and mineral assemblages form a sound basis for deciding on the best compromise.

The objective of this report is to introduce the basic concepts relating mineralogy to the behaviour of an ore and to indicate the value of information, which can be obtained from a correctly oriented mineralogical analysis, in the

development of the initial concepts for the general treatment flowsheet. These can then form the basis for testing by experiment and economic study, leading ultimately to the selection of a specific flowsheet and the design of the ore processing plant.

Sections 1, 2 and 3 are therefore devoted to this theme and are directed more particularly to the less experienced professional to give guidance in setting about a work plan. However, the experienced person should also find these sections useful in putting some of the basic concepts in perspective.

Section 4 is a brief review of the unit operations applied to the typical uranium treatment flowsheets. It includes a summary of the relative energy and cost relationships of the operations. This material is essential to round off the theme of the previous three sections and it acts as a bridge to Part II.

A second objective of this report is to present information, in summary form, of treatment methods applied to various major ore types that are being processed for the production of yellow cake. This is in accord with the following recommendation of the IAEA meeting held in Vienna in 1967 on Processing of Low Grade Uranium Ores [1]:

"To collect and disseminate information on the relative importance of the different ore types for which appropriate methods of treatment have proved successful."

Attention was again drawn to this subject following the 1970 meeting in São Paulo on The Recovery of Uranium [2] and a start was made following the 1975 meeting in Washington on Uranium Ore Processing [3] by the production of a set of tables.

This theme has been expanded in Part II which is in two sections. The first section provides a guide to the characteristics of the major types of uranium ores by way of case studies on specific deposits in various parts of the world (sandstones, conglomerates, vein deposits, granites, pegmatites, carbonatites, syenites, shales, lignites and calcretes.

The second consists of a set of tables summarizing data specific to 76 selected operations. However, Part II also forms an important part of the overall theme of the report, as it forms a nucleus of reference material correlating ore type with treatment characteristics.

1. URANIUM OCCURRENCE AND GEOCHEMISTRY

1.1. Distribution of geochemical cycle

The geochemistry of uranium is largely controlled by the particular characteristics of the element. First, two of the four possible valency states, the four and six valencies, have thermodynamic characteristics such that pH and Eh

conditions in the natural environment can regularly occur, sometimes in adjacent areas, in which one or other valency form is stable. Second, the two resulting ions 4+ and 2+ have very different properties. The highly ionized uranous ion (U^{4+}) has low basicity so it forms an oxide insoluble in dilute acid and its salts hydrolyse readily. Its ionic radius is similar to the lanthanons and to thorium, whose chemistry is also similar as is that of titanium and zirconium so that U^{4+} uranium is often associated with these elements.

Hexavalent uranium occurs as the uranyl ion UO_2^{2+} , the oxide of which is highly soluble. It combines with a number of anions to form soluble complexes in acid and alkaline solutions. It forms compounds with other metal cations and thus forms a wide range of minerals. However, because of the large size of the uranyl ion it does not readily displace nor is it replaced isomorphously, so that its crystalline minerals are generally unique uranium compounds.

Uranium is widely distributed. It is more abundant, for example, than antimony, cadmium, bismuth, iodine and the precious metals. It is not relevant in this report to give an account of the complex story of its distribution; for further details, readers are referred to special geochemical texts, e.g. Rogers and Adams (1969) [4].

The following tabulation gives the order of magnitude (in ppm) of uranium abundance in some rock types:

Mafic igneous rock (basalts, gabbros, etc.):	1
Intermediate igneous rocks (diorites):	2
Acid igneous rocks – grandiorites:	2.6
granites:	4.7
Sediments:	
Normal shales	4
Limestones	2
Black shales	8
	(up to 250)
Seawater:	0.0033

These are mean values, and may have wide limits, especially in black shales which can contain up to 250 ppm and in acid igneous rocks which can contain up to 500 ppm.

Uranium, being of the lithophile elements, has an affinity for silicate (by definition). It is extremely mobile under acidic and alkaline oxidizing conditions, forming the uranyl ion (UO_2^{2+}), which in turn forms a large number of generally complex minerals with a variety of anions and other cations (sulphates, phosphates, carbonates, silicates, vanadates, arsenates and others). The tetravalent form UO_2 (ideal uraninite) is almost insoluble and requires oxidation to the hexavalent form before becoming mobile. However, as pointed out above, conditions are frequently encountered in the natural environment in which hexavalent (uranyl) compounds

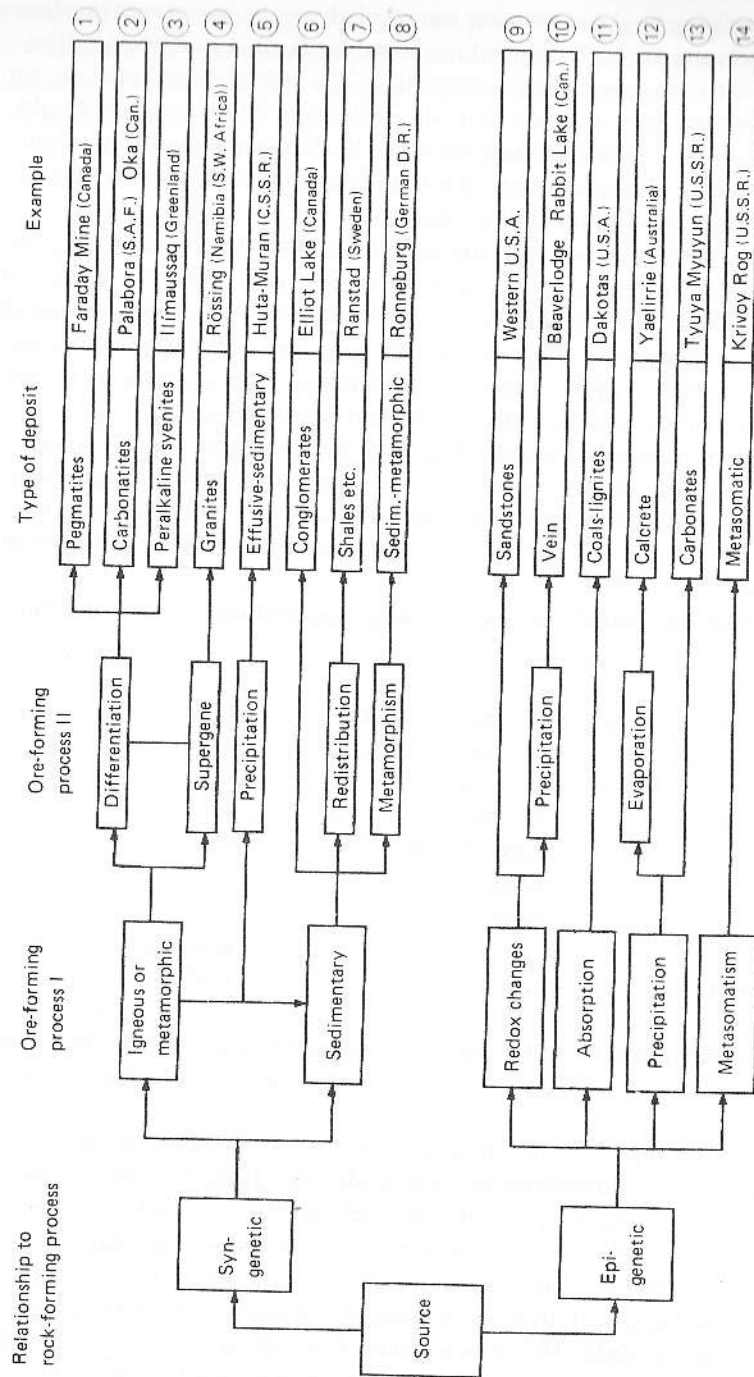


FIG. 1. Genetic types of uranium deposit. (Reproduced from Ruzicka [5].)

are reduced, precipitating UO_2 . This accounts for much of the uranium found in sedimentary deposits and for the high uranium content of some black shales; these contain organic matter which reduces dissolved uranyl compounds, precipitating UO_2 and forming uraninite/pitchblende and uraniferous organic substances.

Uranium occurs in the tetravalent form in a number of primary minerals in igneous rocks and related pegmatites. However, it is more abundant in the minor accessory minerals rather than in the main rock-forming minerals. Such minerals include discrete uranium species (e.g. uraninite, brannerite), as well as species in which uranium is present as a minor component (e.g. zircon, sphene, apatite). These igneous rock sources of primary minerals are seldom economic in themselves (except in the case of pegmatites), and other mechanisms are necessary to generate exploitable deposits.

Such mechanisms include hydrothermal processes, metamorphism, metasomatism, oxidation-solution-reduction-deposition sequences in sediments, precipitation, and other processes as outlined in section 1.2. in the classification.

1.2. Uranium ore deposits

The classification of ore deposits in general, and of uranium deposits in particular, is subject to a number of complications, chiefly because of genetic problems. The mode of formation of many deposits is well understood and thus their classification is unequivocal. However, a number of deposits, including some of the larger ones, require much further investigation and are the subject of extensive discussion. In some cases, agreement regarding genesis may never be reached. The classification of such occurrences is thus problematical.

The use of a form of classification is particularly essential in this report, because of cataloguing the major occurrences required in Part II. (See Part III: Summary Tables on specific uranium deposits and processing plants.)

The classification used here is that of Ruzicka (1975 [5]), which is reproduced in Fig. 1.

A brief explanation of the terms used in Ruzicka's scheme of classification is necessary but for further details the reader is referred to the original paper. The term "Source" is the "original" uranium present in the crust. "Syngenetic" relationships indicate simultaneous or contemporaneous rock- and ore-forming processes; epigenetic relationships indicate that ore-forming processes post-dated already existing rocks.

Ore-forming processes are divided into stages I and II; various combinations of these give rise to fourteen types of deposit, as shown in the final column.

Further subdivision of some types of deposit may be desirable and is certainly possible in some instances (e.g. sandstone deposits). Some occurrences may be categorized into more than one type, i.e. of multiple origin (e.g. some

Australian deposits). Such further subdivision though useful for geological purposes is not required for this paper. More detailed material on ore deposits relevant to their processing is presented in Part II.

1.3. Other uranium occurrences

There are other sources of uranium, in which the element is a very minor constituent and in which it does not generally occur as discrete or specific uranium minerals.

The occurrence of uranium in seawater, in phosphate deposits, in carbonaceous sediments, and porphyry-copper are well known. Uranium in soluble form in groundwater is present in significant amounts in some locations. Primary source rocks (e.g. granites) may become exploitable, particularly in conjunction with processing for other minerals or industrial minerals (micas, feldspars or accessory species such as sulphides, apatite, oxides) and some work was done on the recovery from this source.

2. METALLURGICAL MINERALOGY

2.1. Introduction

The technical characteristics of the process and the economic feasibility of extracting uranium from an ore is mainly influenced by the rock type, characterized by its mineralogical composition and texture.

Among those factors that are influenced by the mineralogy of an ore as distinct from the uranium grade, the following are the most important:

- (a) The degree of comminution required to effect liberation of the uranium mineral;
- (b) The potential for separating the uranium mineral from gangue minerals by physical techniques;
- (c) The nature of lixiviant required (e.g. acid, alkali, oxidant) and the potential level of reagent consumption;
- (d) The probable rheological properties of the minerals in aqueous suspension (with consequent effects on agitation, thickening, filtration, etc.); and
- (e) The probable ionic composition and concentration of the leach liquor (which determine the size of the ion-exchange or solvent-extraction plant and the purity of the final product).

In this report we have defined the term "metallurgical mineralogy" to represent those aspects of the mineralogical investigation that aim at predicting the possible effects of the mineralogy on the above factors. Also, the term

metallurgy encompasses ore dressing, mineral processing and extractive metallurgy. The metallurgical mineralogy of uranium ores does not differ in any basic fashion from that applicable to other ores. However, with a few notable exceptions economic uranium ores are quite low grade, with interest shifting towards even lower grades, a characteristic it shares with gold which, however, does not appear in as many forms as uranium.

The objective of metallurgical mineralogy is ultimately governed by economic considerations, because the extraction and beneficiation process is similarly constrained. Metallurgical mineralogy is quite separate, so far as its technical objective and interpretation is concerned, from geological considerations such as the possible genesis of the ore.

The mineralogist is clearly working in an interdisciplinary field as part of a team, whose objectives are concerned with economic technology not scholastic science. He must develop a facility for communicating his results, in the language of the persons concerned with developing the economic mining of the ore and recovering the metal. It is also clear that he must acquire a basic working knowledge of uranium metallurgy in order to develop that faculty. Mineralogists engaged in this somewhat hazardous no-man's land must enlighten, not blind with science, otherwise their *raison d'être* is lost.

However, the mineralogist also plays a most important role in bridging the area between the economic geologist and the metallurgist, so that his knowledge of the problem must be extended to the orebody in the ground, as well as to the present or proposed processing operations. Therefore, he must become acquainted with the regional geology, the specific geology of the deposit, and the detailed petrology and mineralogy of the immediate host-rocks as well as surrounding or distantly related rocks. Critical appraisal of existing publications and reports is obligatory.

The mineralogist has a formidable armoury of equipment and techniques at his disposal. Although he must be sufficiently familiar with all the methods, to be able to call on them when the problem requires, above all he must develop skill and experience at interpreting what he sees visually. No matter how sophisticated, equipment is no substitute for this skill and any equipment must always be regarded as a tool to be applied in the correct context of the problem.

The mineralogist must keep his mind on the economic objective of the project and resist the temptation to dwell on mineralogical curiosities or to do more than he needs to meet those objectives.

2.2. Uranium minerals

The US Geological Survey has published and revised a glossary of uranium and thorium minerals, the latest revision being in Bulletin 1250 of 1967 (Fron del, Fleischer and Jones 1967 [6]). This Bulletin uses three classifications:

- A. Uranium and thorium minerals (185 listed of which 25 are thorium only).
- B. Minerals with minor amounts of uranium and thorium (55 listed).
- C. Minerals reported to contain uranium and thorium minerals as impurities or intergrowths (20 listed).

No better indication can be given of the rather bewildering number of uranium minerals and the problems caused by nomenclature than to quote from the Bulletin:

"The index of mineral names 534 entries. These represent 260 named species: 185 in section A, 55 in section B, and 20 in section C. Often the question is asked, 'How many uranium and thorium minerals are there?'. If section A is used as the criterion, there are about 185 such minerals to which 300 names have been applied".

A selection of important economic uranium minerals are listed below, with the understanding that this selection is arbitrary since in a local situation a relatively obscure species may in fact constitute a main uranium source. In addition, often the more obscure species may account for much of the uranium in leach tails.

<i>Oxides:</i>	Pitchblende/uraninite, davidite, brannerite, betafite, thorianite, thucholite (partly organic), pyrochlore, microlite, euxenite, gummite, fergusonite;
<i>Silicates:</i>	Coffinite, uranophane, sklodowskite, kasolite, thorite, zircon/cyrtolite, allanite;
<i>Phosphates:</i>	Xenotime, monazite, autunite, torbernite, saleeite, carnotite, sabugalite;
<i>Sulphates:</i>	Johannite, uranopilite;
<i>Carbonates:</i>	Liebigite, bastnäsite, uraniferous calcite;
<i>Molybdates:</i>	Muluranite, umohoite.

2.3. Mineralogical source data

A reference text or handbook of uranium-bearing minerals should contain multiple classifications/listings, based on different parameters, with cross-references. Some of these are as follows:

- (a) Response to u.v. sources (short and long-wave) — a most convenient, rapid and easy technique;
- (b) Colour — a property used almost subconsciously; has limited use but can be effective;

- (c) Density — useful in metallurgical work as well as mineralogical determinations;
- (d) Chemical composition — an indication of U content, and whether this is an essential or non-essential constituent;
- (e) Optical properties — systematic presentation of data based on refractive indices, optic sign; with separate lists of peculiar or distinctive properties (e.g. metatorbernite);
- (f) Crystallography and habit — many uranium minerals tend to be very fine-grained, or altered (metamict), use of crystallography as an aid to identification is limited;
- (g) X-ray diffraction data — there are limitations in that some minerals, particularly those which are the most difficult to identify (the complex oxides), are metamict and non-crystalline. Also, X-ray equipment is not always readily available;
- (h) Solubility in various reagents;
- (i) Mineral associations, both uraniferous and non-uraniferous — these can be useful, providing that the minerals to be associated are in fact genetically related, otherwise the deductions are not valid.

Although a good deal of data remain to be measured on the physical properties of uranium minerals which are most useful for metallurgical interpretation, a number of publications do provide much of this information.

The US Geological Survey Bulletin 1064 (Fron del, 1958 [7]) *Systematic Mineralogy of Uranium and Thorium* contains most of the relevant determinative data or references up to that date. Other references of similar vintage are: Heinrich (1958 [8]) and Getseva and Savel'eva (1956 [9]) and Volborth (1958 [10]).

Bulletin 1064 lists X-ray data, which are also recorded together with more recent data on the card index of the Joint Committee for Powder Diffraction Standards. However, most data since 1958 have appeared scattered through the literature devoted to mineralogy. A table has appeared which provides a quick reference summary to uranium minerals and their properties (Colorado School of Mines Research Institute, 1976 [11]). Quite recently a publication has summarized the minerals and ores of uranium and given many of the properties and diagnostic features [12]. There are also a number of texts on general mineralogy and mineragraphy such as Deer, Howie and Zussman (1962 [13]), Zussman (1976 [14]), and Edwards (1947 [15]).

2.4. Techniques of metallurgical mineralogy

2.4.1. Materials to be examined

If the mineralogist is involved from the beginning of the discovery of an ore deposit, he will naturally progress from the general to the specific details

of that deposit. He will assist in determining the possible causes of geophysical anomalies, will examine outcrop material, percussion samples and drill-core reconnaissance samples delineating geological boundaries, and finally he will determine the mineralogical suitability of available rocks for engineering purposes (concrete for foundations, roads, etc. at the mine site). Thus, some or all the following suites of samples, will be examined in the course of the investigations:

- (i) Regional reconnaissance samples (including stream sediments)
- (ii) Outcrop material from the target area
- (iii) Percussion-drill and/or trench samples
- (iv) Diamond-drill core
- (v) Composite test samples
- (vi) Ancillary material related to specific problems (geophysical, engineering, environmental, mining, safety, etc.).

Much of the material that will be mined first will be from an oxidized zone (if present); thus, its mineralogy is of more immediate relevance and may well be radically different from that of unoxidized ore. However, the largest proportion of the ore reserves is frequently in the zone of unoxidized ore so that emphasis, as far as metallurgy is concerned, can only be decided after the initial assessment of the degree of difference and the evaluation of the ore reserve distribution.

Once the situation is reached where an orebody has been delineated and a viable operation can be considered, the composition of various portions of this orebody (and any others in the vicinity which are to be treated at the same plant) must be determined by systematic sampling, examinations and assays. By these means, major or significant variations which affect subsequent treatment will be detected.

At this stage the work on metallurgical mineralogy is at its most critical, and the following basic data must be determined:

- (i) Spatial variations in mineralogy within an orebody (zonation, transitional ore, the effects of layering, banding, faulting, water tables, palaeogeography, metamorphism and others);
- (ii) Mineralogical variations between different orebodies, (veins, shoots, lenses, lobes, rolls or other structural units);
- (iii) Mineralogical composition of "average" or "representative" samples (as defined and prepared by metallurgists for testing purposes), with emphasis on details relevant to major and common minerals, not the unusual or atypical features. It should be emphasized that such "representative" samples can be misleading because the effects of some factors may be

masked or diluted. Seldom, if ever, are orebodies homogeneous, and thus the feed to a plant constantly varies, although the degree of variation may not always be significant.

- (iv) Mineralogy of special or unusual variations, especially if these are likely to affect processing — such data must be kept in perspective, and their importance assessed, but this information is of prime importance in helping to avoid the problems due to the "masking" effect in the composite sample.

Examination of laboratory, pilot plant and eventually operating mill products will follow, involving concentrates, tailings and various intermediate fractions. Apart from the normal problems of liberation, extraction, recoveries and losses, there should be an awareness of possible economic by-products, whether metallic or non-metallic.

2.4.2. Diagnostic methods

There are techniques which are recognized to form the main armoury of diagnostic methods in mineralogy. The book by Jones and Fleming (1965 [16]) is a useful reference. The following are the basic techniques:

- (i) Examination of thin sections of rocks, chips and grains;
- (ii) Examination of polished sections of grains, chips, millproducts, ore;
- (iii) Examination of grain-mounts, using refractive index oils, of loose material, including stream sediments;
- (iv) Stereobinocular examination of various samples;
- (v) X-ray diffraction (XRD) of hand-picked materials. (The basics of the XRD method require no elaboration. However, the equipment and technique are sufficiently sophisticated to require dedication and specialization for their effective operation.)
- (vi) Electron-Probe Microanalysis (EPMA) has become an invaluable, almost indispensable technique in mineralogy. It suffers two major disadvantages — its restricted availability; and the relatively poor (high) detection limit for uranium (variable depending on instrument and other factors), which may be as much as 0.1% U.

Like XRD the equipment and technique of this method require dedication and especial experience in order to operate effectively. None the less, a basic understanding of the technique is important and readers are referred to Anderson (1973 [17]) in which there is a particular paper, "Application of the electron microprobe in geology", and to Jones and Gavriloni (1966 [18]).

Other techniques of particular relevance to uranium minerals are:

(vii) Ultra violet (u.v.) (short and long-wave) to determine fluorescence (colour and intensity). Many secondary uranium minerals exhibit characteristic fluorescence (Volborth 1958 [10]).

(viii) Autoradiography is a technique specific for the detection of radioactive minerals. It is very useful in determining the distribution of radioactive phases in a sample and it has the advantage of needing little and simple equipment, much of which can be improvised.

The sample is generally either a flat slab of rock, ground but not necessarily highly polished, or a polished section which may consist of grains, chips or solid rock. The smooth surface is placed on to an appropriate piece of film, in a light-tight box, cupboard or darkroom, and left for an arbitrarily chosen time, which depends on the radioactivity level of the sample. Very weakly active samples must be left in contact with the film for up to 10–15 days. Highly active samples require only a few hours. A difficulty appears when a sample may contain both strong and weak sources, since a short exposure time will not cater for a weak source. Two autoradiographs may then be required – the first will involve a short exposure time, revealing strong sources which can then be masked with lead-foil, enabling a second, long exposure to be carried out.

The type of film used is not critical, because of the generally long exposure times involved, but it is better to use a fine-grained film than a high-speed one. X-ray films are very satisfactory, particularly the “occlusal” films used in dental X-ray diagnosis; these are individually packed, ensuring that each sheet is fresh and completely unaffected by stray light.

Generally speaking, autoradiographs are most useful when radioactive sources are difficult to detect, i.e. when discrete U/Th minerals cannot be identified. This occurs most frequently in rocks with low radioactivity distributed in diffuse form.

(ix) Radiometry: In some respects, many radiometric instruments are too sensitive for mineralogical work because the simpler, quicker mineralogical techniques are unable to detect less than approximately 200 ppm U_3O_8 – equivalent to a discrete U mineral in very general terms.

In one laboratory (Fander, 1978 [19]) a small battery-operated Geiger counter (CD-V700 unit) has been found satisfactory. In this instrument the Geiger tube is well shielded, with three narrow slots for admitting radiation. Thus, when passing the detector over a sample, local centres of radioactivity can be quickly pin-pointed if present; alternatively, diffuse activity can be detected if sufficiently strong. If the instrument records no radioactivity, the chances of mineralogical detection are slim, and special techniques must be applied. Thus, the use of such an instrument enables a rapid initial assessment to be made of the mineralogical problems likely to be met, together with an indication of the possible scope and depth of the investigation.

(x) Solution permeability: To measure porosity and permeability of rock strata there are formal and standard methods developed, particularly in the hydrocarbon industry. However, a recent method has been developed in Canada (Kaiman 1977 [20]) for revealing small fractures and planes of penetration for leaching fluids. The method is an adaption of a non-destructive test for fine cracks in metal castings and consists of treating a lump of rock with the dye penetrant “Zyglo” and examining the specimen in ultraviolet light. Sections can be made to show the depth and pattern of the cracks.

(xi) Fission track and neutron activities [21]: The fission tracks developed in a polycarbonate plastic film can be used to detect very low uranium concentrations and determine its distribution in a polished or thin section. The method is superior to autoradiography as it is specific for ^{235}U and has greater sensitivity. Neutron activation of standard samples (in glass) can be used by comparative techniques to obtain a quantitative measure of the uranium present.

2.4.3. Separation techniques

The techniques applied by the mineralogist to separate the various mineral components require special mention. They are designed to produce particular fractions, usually by some concentrating technique that will give a quantitative measure of the amount of mineral. When combined with a controlled grinding procedure a measure of the size distribution is also obtained. Such information is valuable in assessing the potential of physical beneficiation methods and can be a most useful tool in analysing the performance of such methods. The second most useful application is to isolate in a more concentrated form, residual amounts of heavy minerals, which occur in very low concentrations in, for example, tailings – or at least assist in determining whether such fractions exist (as against a ubiquitous distribution through gangue minerals).

However, before dealing with techniques that actually separate the minerals, mention should be made of the important techniques of grain and point counting. In grain counting, the material is spread thinly and the separate grains counted using a microscope (usually binocular microscope). A millimetre grid is used to facilitate the accumulation of the count but avoid counting twice.

In point counting, random “points” are located over the surface and the minerals under the “points” are identified and counted. The random points are generated by automatic movement of a mechanical stage or by an eyepiece graticule.

These are most useful techniques for indicating the proportions of different minerals and, of course, if the mineralogist makes an estimate of grain size, the results give an indication of the size distribution in the particular sample being examined. However, the results will at best represent a very small sample and so far as size is concerned, no account is taken of the breakage characteristics of the ore.

TABLE I. HEAVY LIQUID SEPARATION/ASSAY DATA FOR VARIOUS SIZE FRACTIONS OF A URANIUM ORE COMPOSED OF QUARTZ (DENSITY 2.65 g/ml), CHLORITE (DENSITY 2.9 g/ml), URANINITE (DENSITY 9.5 g/ml), SULPHIDES AND MAGNETITE (DENSITY 5–7.5 g/ml)

Size fraction (mm)	Density (g/ml)	Weight (%)	U ₃ O ₈	
			Assay (%)	Distribution (%)
+ 1.68	< 2.8	98.4	0.42	52.9
	2.8–4.2	1.6	23.0	47.1
	> 4.2	—	—	—
		100.0	(0.78)	100.0
– 1.68 + 0.85	< 2.8	97.8	0.37	45.5
	2.8–4.2	1.9	20.0	47.8
	> 4.2	0.1	53.0	6.7
		100.0	(0.80)	100.0
– 0.85 + 0.42	< 2.8	96.9	0.30	34.7
	2.8–4.2	2.5	9.0	26.9
	> 4.2	0.6	53.6	38.4
		100.0	(0.83)	100.0
– 0.42 + 0.21	< 2.8	95.2	0.11	12.0
	2.8–4.2	3.5	3.32	13.4
	> 4.2	1.2	54.0	74.6
		100.0	(0.87)	100.0
– 0.21 + 0.105	< 2.8	93.5	0.06	4.8
	2.8–4.2	4.5	1.44	5.2
	> 4.2	2.0	55.4	90.0
		100.0	(1.25)	100.0
– 0.105 + 0.053	< 2.8	91.3	0.05	2.8
	2.8–4.2	6.0	1.09	4.0
	> 4.2	2.7	56.7	93.2
		100.0	(1.64)	100.0

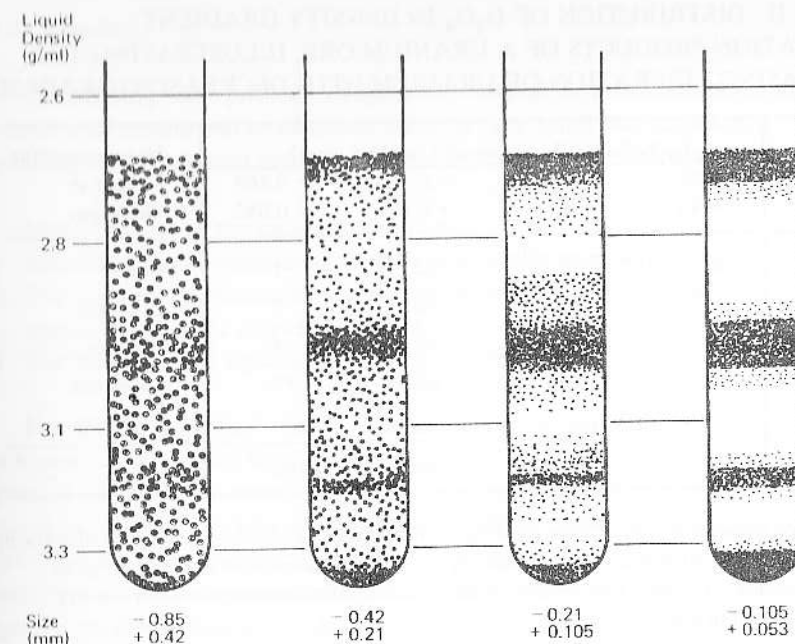


FIG. 2. Density gradient separation of size fractions of a uranium ore illustrating the increasing mineral liberation with decreasing size.

The methods used for physical separation are based on the same properties applied by the metallurgist. Methods generally applied (Zussman 1967 [14]) take advantage of differences in specific gravity, magnetic, electrical conductivity and surface properties, the most widely applied being the first two listed. For specific analytical purposes hand picking is also used.

Gravity methods: The Haultain superpanner, micro-panners, wet tables (such as Wilfley) and, for some purposes, elutriators, are most useful for concentrating heavy minerals and closely simulate metallurgical operations. For the best analytical data on liberation, the use of heavy liquids is indispensable. There are two main techniques.

In the first method, the density of the separating medium is adjusted so that it lies between the specific gravity of the minerals to be separated. The technique is most useful for demonstrating the increasing liberation of mineral from composites as the grind becomes finer. Table I, taken from Henley et al. (1972 [22]), illustrates the results of such a technique. When other heavy minerals are present the technique can be combined with microscope techniques such as area estimation (Gaudin 1939 [23]) and point counting.

TABLE II. DISTRIBUTION OF U_3O_8 IN DENSITY GRADIENT SEPARATION PRODUCTS OF A URANIUM ORE, ILLUSTRATING THE INCREASING LIBERATION OF URANIUM WITH DECREASING GRAIN SIZE

Density product (g/ml)	U_3O_8 distn. (%) within size fraction (mm)				Main mineral(s) present at liberation
	- 0.85 + 0.42	- 0.42 + 0.21	- 0.21 + 0.105	- 0.105 + 0.053	
2.6-2.8	24	8	2	1	Quartz
2.8-3.1	36	34	20	3	Dolomite
3.1-3.3	30	36	16	4	Fluorite
> 3.3	10	22	62	92	Sulphides Uraninite
	100	100	100	100	

The data presented in this Table show that with decreasing size (and increasing liberation) progressively more of the uranium within a size fraction concentrates in the > 3.3 g/ml density product. The data may be compared with the visual appearance of the density gradients shown in Fig.1.

The second heavy liquid technique uses a density gradient column (Muller and Burton 1965 [24]) and is best applied to each size fraction of the screened ore. In this technique the minerals separate into their various density positions in the liquid column. They tend to concentrate in bands, which sharpen as the grind becomes finer, giving fewer composite particles. The bands can be isolated and the minerals described and their quantities measured. Figure 2 and Table II illustrate the results of the technique (Henley et al. 1972 [22]).

Magnetic methods: Magnetic susceptibility data for uranium minerals are scanty as noted by Hartmen and Wyman (1969 [25]). However, the application of magnetic separation techniques can be useful in empirically determining whether the uranium is associated with the magnetic or non-magnetic fractions. Techniques include the use of the Frantz Isodynamic Separator and high-intensity wet separators such as the Davis Tube.

The various separation methods are the same as those applied to any other mineralogical problem which must be pursued in detail. Thus, supporting chemical analyses are often required, especially in those cases where the uranium is present in some form of submicroscopic inclusions or in substitution.

2.4.4. Results

By applying these procedures (generally several, in series and in parallel), the following results will be obtained with varying detail and accuracy, depending on the inherent nature of the ore, and the depth/extent of the investigation.

- (i) Identification of the radioactive phases;
- (ii) Identification of other economic or potentially economic phases;
- (iii) The textural relationships, grain sizes, distribution, paragenesis, and associations of the phases identified under (i) and (ii);
- (iv) The metallurgical significance of the observations obtained.

For the mineralogical results to be of maximum benefit to the metallurgist, it is necessary to present them in logical sequence, clearly set out, with the minimum of geological terms, and incorporating observations relevant to processing. A good method of presenting results is by listing each mineral, in order of economic importance, processing significance and/or abundance. All mineralogical data (as mentioned above) should be presented together with emphasis on special features (e.g. iron-staining, microfracturing or other details likely to affect processing). Generally too, it is helpful to quote mineral formula, density and Moh's hardness, uranium content if applicable, and other details of value to the metallurgist.

Specific information concerning chemical reactivity, possible sliming, possible flotation problems, behaviour of the material in grinding, anomalous magnetic features, shape factors, undue tarnishing effects and other details of this nature form the bridge between mineralogy and metallurgy. Wherever the mineralogist can foresee potential problems, he should point these out as early as possible.

2.5. Suggested modus operandi

Let us assume that suites of samples are received in the laboratory for mineralogical examination and comment. The suites at various times may comprise the following:

- (1) Percussion chip samples of weathered and fresh material;
- (2) Drill core intersections of mineralized and unmineralized rocks;
- (3) Metallurgical test products, including leach residues.

Examination of these three samples will generally proceed separately because they represent different stages of the investigation of a deposit. Thus, the suggested procedures will be listed in sequence, although there is bound to

be some overlap. It should be emphasized that observations pertinent to metallurgy should be made as early as possible in the whole project, even though these may have to be modified later, as more data accumulates. Since, in most cases the weathered portion of the deposit will be mined first, its mineralogical characteristics and the differences between this and the fresh ore must be emphasized, to aid thinking and planning ahead.

2.5.1. Percussion-chip samples

Percussion-chip samples are generally produced early in the evaluation of a potential ore deposit, during drilling operations. Therefore, the mineralogist is likely to base the first information to the metallurgist on the basis of such samples. Because of the nature of percussion-chip samples, in which the particles seldom exceed 10–15 mm in size and range down to very fine (“slime”) material, the continuity, or fabric, of the rocks has been disrupted. For this reason, percussion-chips yield only limited information; this particularly applies to coarse-grained rocks (conglomerates, granites). Also, soft components may be lost or their proportions much reduced.

Thus, percussion-chip samples must be treated with caution in terms of the data produced and examination of further material (drill-core, specimens from test-pits, shafts or trenches) is essential for obtaining a more complete picture of the occurrence. However, they have the advantage of being more representative than hand-specimens.

The examination of percussion-chip samples could take the following course:

- (i) Radiometric and ultraviolet scanning, in conjunction with stereobinocular microscope (low power) examination;
- (ii) Selection of fluorescent material and most radioactive chips, for separate identification of radioactive phases;
- (iii) Preparation of thin and polished sections.

Frequently it is necessary to prepare thin-sections of friable, weathered rocks, porous material, rock-chips or grains. A technique is outlined below which may be used quite successfully, without resorting to vacuum-impregnation or embedding methods.

Prepare a stock of small slabs of brick (ordinary clay bricks), each about 25 X 50 mm in size and 5–10 mm thick;

Place a brick slab on a hot plate and place the sample, or fragments or gains, on the slab, as an even layer of closely-packed particles;

Allow the slab plus sample to warm through thoroughly, expelling moisture, but do not overheat;

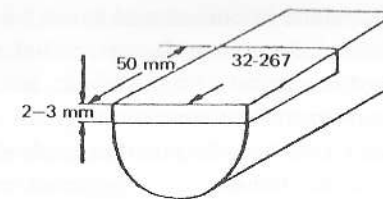


FIG.3. Sampling half-core (reproduced from Kitagawa [26]).

Impregnate the sample (and part of the slab) with an epoxy resin (of low viscosity if obtainable, with a long setting time), so that all particles are thoroughly wetted and covered, forming a cohesive mass;

Allow the sample to set on the warm hot plate over a period of several hours; the process should not be too speedy, otherwise impregnation will be incomplete. For this reason a slow-setting resin is essential;

When the whole sample (+ slab) is thoroughly set, it is ground flat like a normal rock slab, and is then cemented in the usual way to a microslide;

Subsequent preparation is the same as for a normal rock section.

- (iv) Preparation of microphotographs and autoradiographs of polished sections. Study of both together to establish nature and distribution of radioactivity;
- (v) Examination of all sections to determine the uranium and other minerals;
- (vi) Separation by hand-picking or other methods of uranium minerals for further identification/confirmation by XRD, EPMA and chemical/spectrography analyses.

2.5.2. Drill core samples

Solid core samples, if examined in sufficient number to be representative of lateral and vertical and other variations, should provide adequate data for the metallurgist. They have the additional advantage of giving useful geological information for correlation, interpretation of genesis, and exploration.

A sketch of how samples for mineralogical examination can be obtained from half core is shown in Fig.3.

Each sample should consist of a slice of core about 50 mm long and 2–3 mm thick, taken longitudinally at predetermined intervals over the entire length of the half core. Each slice of core should be marked with an arrow as shown in Fig.3 to indicate the direction of the bottom of the borehole, as this may be helpful to the mineralogist in studies related to the geology of the orebody. The number of the borehole and the depth of intersection (e.g. 32–267 Borehole 32; depth 267 m) should also be marked on each sample slice as illustrated in Fig.3 for identification.

The frequency at which slices of core should be cut for microscopic work from each borehole will depend upon many factors, including the total length of mineralized intersection, the number of types of ore, and the intersected length of each of these ore types in each borehole.

There are no fixed rules as to how frequently sample slices should be cut from core for microscopic work; however, one suggested criterion is tabulated as follows:

Length of intersection	1 m	2 m	3 m	4 m	5–20 m	> 20 m
Number of sample slices	1	2	3	4	5	10

It is important to document exactly where each slice of core was cut, whatever criterion is selected for taking samples, so that the mineralogical data can be related to the overall orebody.

The types of examination are the same as for percussion chip samples. In addition, the offcuts from thin-section preparation are useful for preparing autoradiographs, which may be directly compared with the thin sections; this is invaluable in studying non-opaque radioactive phases, especially where uranium and thorium are in solid solution or some other concealed form (e.g. phosphates, clays, other silicates, sulphates etc.).

Apart from the advantages of studying coherent masses of core in order to obtain data on the continuity, distribution and interrelationships of the constituents, long sections or slabs of core may be ground and polished (or varnished), etched, autoradiographed, stained, or subjected to other techniques for specialized study on a larger, semi-macroscopic scale; this provides a useful bridge between microscopic examinations and field observations.

2.5.3. Metallurgical test products

Since the great majority of metallurgical samples are more or less finely ground, the information obtained is almost entirely mineralogical, and is, in a sense, restricted, for instance textural relationships are confined to the dimensions of the grains examined.

Thin sections may be prepared of coarser fractions (say + 200 μm , depending on other factors such as general grain size), especially to study the nature of composite grains, non-opaque inclusions, and other details.

Often, however, non-opaque minerals are determined by preparing grain-mounts on glass slides. Mounting media may be permanent (e.g. epoxy resins, Canada balsam, Lakeside or similar preparations), or they may be temporary, using refractive index oils. The latter is generally the preferred technique, because an appropriate oil can be chosen, and because single grains can be picked out or a different oil can be used on the same grain-mount. For instance, the

use of methylene iodide (di-iodo-methane) will automatically classify the grains into those with a refractive index greater or smaller than 1.74, and with a density above or below 3.3 g/cm³, if grain-mounts are so prepared that most are able to sink or float. Thus, such mounts have multiple purposes, yielding additional information. It may be necessary to de-slime samples carefully in order to clean individual grains before they can be identified; needless to say this operation must be carried out with extreme care, to minimize losses of "light" constituents. If necessary, the "slimes" can be dried and examined separately (e.g. by XRD for determination of clays).

Opaque phases are examined in polished sections; such sections can also be used for autoradiography or subsequent EPMA. The main difficulty is that polished sections of grains, especially mixtures of minerals with a wide range of densities, are seldom representative. This difficulty is not at all easy to overcome, and it is safest to use polished sections of grains only for qualitative mineralogical determinations, or for relative assessment of individual opaque phases, using a grain-mount for determining total number of opaque grains.

When examining polished sections of grains, in which the plane of the section makes arbitrary intersections with the individual grains, it should be remembered that "composite" grains are always more abundant than can be observed, by a factor which varies with the ore deposit. This is because a certain number of "composite" grains are intersected in such a way as to show only one component in the polished surface; this bias works in only one direction (free grains are always free, composites may appear free, depending on the intersection).

Of course, in particular it is to metallurgical test products that the results of the separation techniques described under 2.4.3 are applied. Liberation studies on screen fractions and the isolation of trace amounts of heavy minerals from tailings are two examples.

2.6. Conclusions

It is well within the scope and competence of reasonably equipped and experienced mineralogists to obtain much information for the metallurgist to plan his programme and guide his experimental strategy.

It is essential that all personnel involved in the exploitation of a deposit are aware that such information can be obtained, and to be conscious of its usefulness if properly handled and presented.

The mineralogical information should be obtained at the beginning, not half-way through a project or, as is far too commonly the case, after spending fruitless effort using conventional techniques of metallurgy on a trial and error basis, when a more rational and sometimes innovative plan of attack could be formulated by prior knowledge of mineralogy.

3. URANIUM ORE PROCESSING METALLURGY

3.1. Introduction

Processing of uranium ores for the production of a concentrate has been effected predominantly by the application of chemical hydrometallurgy. This fact contrasts for example with the production of the common base metal concentrates, which are produced predominantly by physical beneficiation. With a few notable exceptions physical beneficiation has not been applied for concentrating uranium and when it has been used, the beneficiation process has improved economic performance but its application has not been mandatory for achieving economic viability. Three exceptions are Radium Hill, Armstrong (1965, [27]); Palabora, Tunley [28]; Pendreigh and Ergo (1978, [29]).

Several contributing factors have relegated physical beneficiation to a minor role. The main ones relate to mineralogy. Thus, in the ores from which the major amount of uranium concentrate has been derived the following applies:

(a) Uranium occurs in minerals from which it can be taken into solution by chemical means with a high degree of selectivity from its associated gangue and with good recovery.

A low-grade ore which does not meet this criterion will almost certainly require physical beneficiation (as the three exceptions quoted above have demonstrated).

(b) The uranium minerals occur in the intergranular matrix generally as fine to very fine grains and often widely dispersed in the fine matrix minerals.

Although this property has enabled a limited application of size classification, in general it is a condition unfavourable to physical concentration methods which are notoriously ineffective in treating very fine material.

(c) A third factor common to all uranium ores concerns grades and value. Compared, for example, with base metals the grade of uranium ore feed is very low. Thus, compared with copper ores considered to be low grade (the big porphyry deposits) uranium ores at 0.05 to 0.10% are an order of magnitude less, and the volume percentage of the minerals often much less. To achieve a significant concentration increase with high recovery from such a grade by physical methods is a formidable problem.

These are some of the reasons why chemical metallurgy is still the major route to processing. It is apparent that the situation may change as the grades of traditional sources decline and as more sources are examined of low-grade refractory ores. However, if the basic characteristics of ores remain the same, we may well be faced with accepting lower recoveries (as in the case with alluvial tin recovery) and/or require the development of new technology to meet this challenge.

3.2. Processing chemistry

3.2.1. Chemical response of uranium minerals

Uranium minerals can be dissolved by acid solutions or alkaline (carbonate) solutions.

There are two valency states in which uranium occurs naturally — the hexavalent form, the oxide of which is UO_3 , and the tetravalent UO_2 . In its hexavalent form uranium goes directly into solution as illustrated by Eq.(1).



In the case of hexavalent salt-type mineral compounds, a simple double decomposition applies by thermodynamic solubility equilibria criteria. In solution, the uranyl ion forms complexes with the sulphate ion and in the case of alkaline solutions with the carbonate ion, thus:



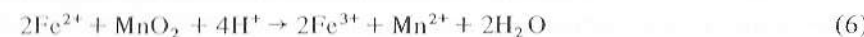
For reaction (1), as applied to the alkaline carbonate process, the hydrogen ion is supplied by bicarbonate which must be present for this purpose. When the uranium is present in the tetravalent form as in UO_2 , such as it is in part in uraninite, it does not dissolve at perceptible rates and requires oxidizing to the hexavalent form; thus:



The oxidation reaction can be quite complex, rapid oxidation in acid medium being achieved mainly by the presence of ferric ions in solution:



Hydrogen ions are not involved in this reaction. However, to maintain the dissolution of the UO_2 , the Fe^{3+} must be renewed by subsequent oxidation of the Fe^{2+} formed in Eq.(5), a reaction which requires hydrogen ions. Thus, if manganese dioxide is used as the source oxidant, the following applies:



The role of the ferric ion in the acid dissolution of UO_2 is most important. It acts essentially as an electron transfer "catalyst". Neither oxygen, manganese dioxide nor chlorate ions, for example, is effective in oxidizing UO_2 at practical rates under ambient conditions of temperature and pressure.

Ferric ions cannot be maintained in alkaline carbonate solutions and it is the lack of such a catalyst which is largely responsible for the very different conditions required in carbonate leaching as compared with acid leaching. Carbonate leaching calls for more severe conditions of pressure and temperature, often a longer leaching time, and finer grind.

This is a simplified summary of the basic chemistry of the leaching processes. For greater detail refer to Merritt (1971 [30]), Clegg and Foley (1958 [31]), Wilkinson (1962 [32]) and Laxen (1973 [33]). However, although it is important to understand the stoichiometry of this basic chemistry, the economics of the leaching process are controlled by a complex interaction of kinetics and stoichiometry. Thus (in the absence of certain anions such as PO_4 , VO_4 etc.), uranium will stay in solution at pH 3.0 to 3.5 and indeed a fresh precipitate of uranium can be redissolved at pH 3.0. However, the rate at which almost any uranium mineral will dissolve at ambient temperatures is too low at pH 3.0 for practical industrial extraction processes. The rate of solution depends upon those parameters which affect the kinetics of heterogeneous chemical reactions generally, namely reagent concentration, temperature, particle size and mechanical processes affecting diffusion (e.g. agitation).

However, given a fixed set of conditions, the reactivity of the minerals then plays the major role in the rate process. Fortunately, many of the commonly occurring uranium minerals are relatively easy to leach. More important, they generally leach under conditions in which most of the associated gangue minerals are relatively unreactive. There are some notable exceptions and a number of uranium minerals which are so refractory to acid attack that they cannot be treated economically without first separating them from the major portion of their more reactive host-rock minerals. Those minerals most refractory to acid attack tend to have their uranium in the four-valent state (although many four-valent uranium minerals are not refractory provided suitable oxidation conditions are provided) and alkaline carbonates are generally ineffective in dissolving them.

The major part of the world's economic uranium resources exist in uraninite or pitchblende and, although the rate of dissolution will vary depending upon the form, grain size, and association with host minerals, in general a satisfactory dissolution rate can be effected with sulphuric acid solutions of pH 1.5 to 2.0 and alkaline carbonate solutions containing between 30 and 60 g/litre Na_2CO_3 and 5 to 15 g/litre NaHCO_3 , provided suitable oxidation conditions are maintained. In general, similar conditions will dissolve the hydrated oxides such as gummite, shoebite and becquerelite as well as hydrated minerals and some hydro-silicates such as coffinite and uranophane.

Uranium minerals containing anions of phosphorus and vanadium are not uncommon. Thermodynamic equilibrium solubility considerations alone suggest that they require relatively higher acidities to be maintained in the leaching and subsequent leach solution. If torbenite and autunite are major contributors of the uranium in an ore, pHs of 1.5 to 1.0 are more effective in achieving speedy dissolution.

Reactivity of mixed-oxide minerals and combinations with titanium and zirconium are significantly less than the simple uranium oxides and salt compounds. Thus, brannerite, an important source of uranium, generally requires acidities of at least 0.5N sulphuric acid. However, their reactivity can vary greatly; some uraniferous zircons, for example, will readily dissolve at these concentrations but others are most refractory. Davidite also is quite refractory. The only commercial operation which treated this mineral used boiling 7N sulphuric acid for the initial leaching.

3.2.2. Chemical response of gangue minerals

The reactivity of the gangue minerals and the contribution that their dissolution products make to the leach solution have profound effects on the chemistry and economics of uranium extraction.

Quartz: Quartz is a non-reactive mineral and the fact that it is the major component in some of the world's largest uranium occurrences, such as the conglomerates of South Africa and Canada, contributes (by default of other reactive minerals) in no small measure to the economic exploitability of these ores.

Carbonates: Although some dolomites and siderites react relatively slowly, carbonate minerals in general consume their stoichiometric equivalent of acid at the pH required for the dissolution of uranium minerals, and their presence as a major component has usually been the determining factor in initiating the consideration of alkaline carbonate leaching. Gypsum by contrast is a problem in the alkaline circuit, reacting to form calcium carbonate and forming sodium sulphate in solution.

Phosphates: Apatite is not as reactive in acid as carbonate and its reactivity is variable. However, to a greater or lesser degree, it consumes acid at relatively low acidities and its rate of dissolution at pH 1.5 or less is significant. In addition, the phosphate going into solution can complex the ferric ions and thus inhibit their role in the oxidation process. Another effect is that, in subsequent processing, phosphate ions can cause reprecipitation of uranium if the pH is not maintained below 2.0. Finally, some phosphate may find its way to the yellow cake in unacceptable amounts.

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This is a simplified summary of the basic chemistry of the leaching processes. For greater detail refer to Merritt (1971 [30]), Clegg and Foley (1958 [31]), Wilkinson (1962 [32]) and Laxen (1973 [33]). However, although it is important to understand the stoichiometry of this basic chemistry, the economics of the leaching process are controlled by a complex interaction of kinetics and stoichiometry. Thus (in the absence of certain anions such as PO_4 , VO_4 etc.), uranium will stay in solution at pH 3.0 to 3.5 and indeed a fresh precipitate of uranium can be redissolved at pH 3.0. However, the rate at which almost any uranium mineral will dissolve at ambient temperatures is too low at pH 3.0 for practical industrial extraction processes. The rate of solution depends upon those parameters which affect the kinetics of heterogeneous chemical reactions generally, namely reagent concentration, temperature, particle size and mechanical processes affecting diffusion (e.g. agitation).

However, given a fixed set of conditions, the reactivity of the minerals then plays the major role in the rate process. Fortunately, many of the commonly occurring uranium minerals are relatively easy to leach. More important, they generally leach under conditions in which most of the associated gangue minerals are relatively unreactive. There are some notable exceptions and a number of uranium minerals which are so refractory to acid attack that they cannot be treated economically without first separating them from the major portion of their more reactive host-rock minerals. Those minerals most refractory to acid attack tend to have their uranium in the four-valent state (although many four-valent uranium minerals are not refractory provided suitable oxidation conditions are provided) and alkaline carbonates are generally ineffective in dissolving them.

The major part of the world's economic uranium resources exist in uraninite or pitchblende and, although the rate of dissolution will vary depending upon the form, grain size, and association with host minerals, in general a satisfactory dissolution rate can be effected with sulphuric acid solutions of pH 1.5 to 2.0 and alkaline carbonate solutions containing between 30 and 60 g/litre Na_2CO_3 and 5 to 15 g/litre NaHCO_3 , provided suitable oxidation conditions are maintained. In general, similar conditions will dissolve the hydrated oxides such as gummite, shoopite and becquerelite as well as hydrated minerals and some hydro-silicates such as coffinite and uranophane.

Uranium minerals containing anions of phosphorus and vanadium are not uncommon. Thermodynamic equilibrium solubility considerations alone suggest that they require relatively higher acidities to be maintained in the leaching and subsequent leach solution. If torbenite and autunite are major contributors of the uranium in an ore, pHs of 1.5 to 1.0 are more effective in achieving speedy dissolution.

Reactivity of mixed-oxide minerals and combinations with titanium and zirconium are significantly less than the simple uranium oxides and salt compounds. Thus, brannerite, an important source of uranium, generally requires acidities of at least 0.5N sulphuric acid. However, their reactivity can vary greatly; some uraniferous zircons, for example, will readily dissolve at these concentrations but others are most refractory. Davidite also is quite refractory. The only commercial operation which treated this mineral used boiling 7N sulphuric acid for the initial leaching.

3.2.2. Chemical response of gangue minerals

The reactivity of the gangue minerals and the contribution that their dissolution products make to the leach solution have profound effects on the chemistry and economics of uranium extraction.

Quartz: Quartz is a non-reactive mineral and the fact that it is the major component in some of the world's largest uranium occurrences, such as the conglomerates of South Africa and Canada, contributes (by default of other reactive minerals) in no small measure to the economic exploitability of these ores.

Carbonates: Although some dolomites and siderites react relatively slowly, carbonate minerals in general consume their stoichiometric equivalent of acid at the pH required for the dissolution of uranium minerals, and their presence as a major component has usually been the determining factor in initiating the consideration of alkaline carbonate leaching. Gypsum by contrast is a problem in the alkaline circuit, reacting to form calcium carbonate and forming sodium sulphate in solution.

Phosphates: Apatite is not as reactive in acid as carbonate and its reactivity is variable. However, to a greater or lesser degree, it consumes acid at relatively low acidities and its rate of dissolution at pH 1.5 or less is significant. In addition, the phosphate going into solution can complex the ferric ions and thus inhibit their role in the oxidation process. Another effect is that, in subsequent processing, phosphate ions can cause reprecipitation of uranium if the pH is not maintained below 2.0. Finally, some phosphate may find its way to the yellow cake in unacceptable amounts.

Silicates: Of the silicate minerals little systematic work has been done to indicate a particular order of reactivity. The Bowen's Reaction Series can give some indication. Certainly, of the "primary" and unweathered rock-forming minerals, it is reasonable to expect that the more basic the mineral the more reactive to acid it might be. This is illustrated in the reactivity of the allanite host mineral in the Mary Kathleen ore of Australia.

However, the silicate minerals occurring in most of the world's large uranium deposits are those associated with weathered material in sedimentary rocks, in metasediments and their weathering products.

It is these minerals — biotite, chlorite, sericite and various clay minerals and their various proportions — which in the absence of highly reactive minerals, tend to effect the greatest control in the treatment process, both leaching and subsequent operations. All of them react in acid to a greater or lesser extent at pH below 2.0 and can be particularly reactive below pH 1.5. The final choice of acidity, time and temperature of leach, to optimize the recovery and economic value of the recovery, is to a large extent controlled by the differential rate of attack on these ore minerals as compared with the uranium. These optimum conditions can only be determined by experimental testing procedures.

Iron oxides: The reactivity of the various iron oxides and hydrous oxides is also variable but the rate of dissolution is usually significant at the pH range used in acid leaching of uranium ores. In fact these minerals and the ferromagnesium minerals, present to a greater or lesser extent in all ores, contribute the all-important ferric ions for the oxidation reaction.

Sulphides: The behaviour of sulphides like the silicates cannot be explicitly predicted. However, in general the presence of sulphides suggests that more oxidants may be required and as a result also possible higher acid consumption. In addition, their presence may indicate that the ore may be subject to rapid weathering and bacterial leaching in stock-piles etc. In fact the presence of pyrite in low-grade ores should suggest that bacterial leaching be examined.

Carbonaceous constituents: Graphite and related materials present in carbonaceous shales are generally unreactive but can lead to problems owing to the physical locking of the disseminated uranium mineral, or sometimes in the solid liquid separation processes.

3.2.3. Chemistry of concentration from solution

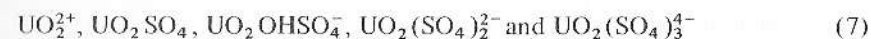
The solutions obtained from leaching the ore contain a complex mixture of cations and anions. If alkaline leaching is used, an advantage is demonstrated in the fact that the leaching process effects a more selective separation. How-

ever, the uranium concentration is still quite small, so that concentration and/or recovery processes must still be applied.

From the acid leaching processes, the composition of the resulting solution depends on mineralogy, but there is always a proportion of aluminium, iron and magnesium and usually some silica and a variety of other metals such as titanium, vanadium, lanthanons and thorium.

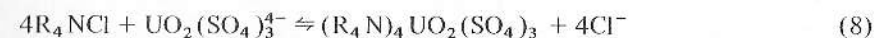
At concentrations of between about 0.5 and 1.5 g/litre, uranium is therefore a minor constituent, considering the molecular weight involved. Thus, the molar concentration of UO_2^{2+} is seldom more than 0.005 in solutions which are typically greater than 0.15 molar in total sulphates, up to 0.3 molar being not uncommon. A highly selective process is therefore required to prepare a high-grade uranium product from such solutions.

Three main processes have been used — selective direct precipitation, ion exchange, and solvent extraction. The following species can be present in the sulphate solution:

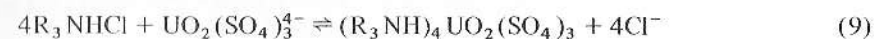


In the range of concentration and pH usually existing in process solutions, $\text{UO}_2(\text{SO}_4)_3^{4-}$ is the dominant species. The characteristics of these ions, such as multivalence and low solvation, favour the adsorbate in ion-exchangeable systems, and advantage is taken of this in the use of solid ion exchangers (resin) and liquid ion exchangers (solvent). General reactions of the following type take place:

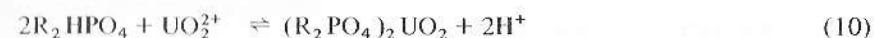
- (1) Quaternary amine — anion exchange: most resins are of this type, e.g. resin in chloride form:



- (2) Tertiary amine — anion exchange: most solvent systems use this type, e.g. a solvent in a chloride form:



- (3) Alkyl phosphate — cation exchange: some solvent systems are of this type:



No cation exchange system has been used with resins. In industry, the resin system is called ion exchange (IX) and the liquid solvent system is called solvent extraction (SX) and will be referred to thus.

Ion-exchange processing is also applied to the extraction and concentration of uranium from alkaline sodium or ammonium carbonate solutions. The predominant anion species in this case is $\text{UO}_2(\text{CO}_3)_3^{4-}$ which undergoes exchange by mechanisms similar to those shown for the complex sulphate anions.

Both IX and SX consist of two parts. In the first part, liquor is contacted with organic phase to transfer the uranium relatively selectively into the organic phase. This is termed the absorption stage in IX and the extraction stage in the SX. In the second part, the organic phase is contacted with an aqueous solution of such a nature that it strips the uranium back into a high-grade, relatively pure, uranium solution. In IX this is termed the elution, and in SX the stripping stage.

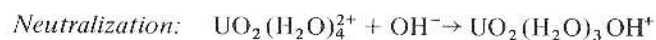
These processes result in a considerable concentration of the uranium, the final solutions to precipitation containing between 15 and 30 g/litre U_3O_8 . However, although they are fortunately highly selective, there are a number of elements which can be derived from the minerals in the original ore and cause problems in these processes, owing to relatively irreversible reactions causing "poisoning" or fouling, or owing to the fact that they pass through the processes to a sufficient degree to appear as impurities in the final product. Some such elements are silicon, iron, titanium, thorium, lanthanons, molybdenum, phosphorus and vanadium.

3.2.4. Chemistry of precipitation

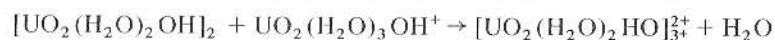
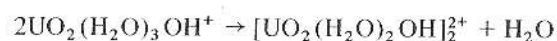
The final chemical stage of the acid process is the precipitation from the high-grade solution. Although direct reduction to UO_2 and precipitation as uranium tetrafluoride and also as peroxide have been proposed and studied, commercial practice uses precipitation by pH adjustment.

The precipitation process is quite complex and much remains to be elucidated. Essentially it consists of converting the basic UO_2^{2+} cation into an insoluble precipitate which can contain a mixture of the compounds: hydrated uranium oxides, basic sulphates and uranates.

The basic UO_2^{2+} ion has a linear structure and combines with water to form an octahedrally co-ordinated complex ion $\text{UO}_2(\text{H}_2\text{O})_4^{2+}$. As the pH is increased the following types of reactions are thought to occur:

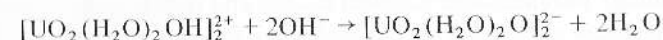


Polymerization by condensation: e.g.



Anion combination: This can take the form of more OH^- neutralizing the polymeric cations and precipitating as hydrated oxide or partial combination with SO_4^{2-} to precipitate basic sulphate.

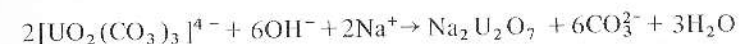
Diuranate ions form probably because of the attack on the hydrogen of the OH bridge in the polymeric cation by free OH^- ions, thus:



With this complex chain of simultaneous reactions it is little surprise that the physical and chemical properties of the precipitate are strongly dependent on the conditions such as type of base employed; rate of addition of base; agitation conditions; whether the process is interrupted (held for any time at a given pH); and the temperature used.

The precipitation from alkaline leach liquor is no doubt equally complex. Acid can be added to neutralize fully the basic anions to pH 6.0 when the uranium precipitates mainly as a hydrated oxide with some diuranate. However, to recover the alkaline reagent, precipitation is usually effected by addition of caustic soda. The free HCO_3^- ions are first neutralized and then the carbonate complex dissociated.

However, in the high pH condition existing, the final product formed is predominantly the diuranate and the overall reaction can be represented by



None of the above is particularly dependent on mineralogy. However, as pointed out earlier, although the concentrating processes of IX and SX are selective, certain metal ions can "leak" through to the concentrated solutions. In the ion-exchange process, in particular, ferric ions can appear in significant concentration and this has led to the split precipitation procedure at pH 3.0 to 3.5, sometimes using a cheap base such as lime. The uranium is then recovered from the filtrate of the iron precipitate.

Most other metal impurities derived from the original minerals tend to be controlled by introduction of fractional elution or scrubbing procedures in the IX or SX circuit (e.g. thorium, rare earths, molybdenum).

3.3. The metallurgical investigation

3.3.1. Development of a project

The four main phases through which a project may pass are as follows:

- (a) *Prospecting:* This phase generally covers work leading up to the discovery of target areas for exploration. Ad hoc samples are analysed and investi-

gated mineralogically so that a preliminary idea can be obtained of the amenability of the ore to processing.

- (b) *Exploration into evaluation:* During this stage the presence of an orebody is confirmed and its average grade, ore reserves and geology are determined by further drilling. Detailed mineralogical studies and bench-scale metallurgical tests are conducted on borehole-core samples and a preliminary evaluation of the flowsheet and economics are undertaken.
- (c) *Development and basic engineering:* This phase may include trial-mining operations, pilot-plant proving of the selected process on bulk samples, compilation of the basic engineering specifications for mining and ore-processing equipment, preparation of capital and operating cost estimates and the undertaking of a full feasibility study.
- (d) *Detailed engineering, construction and commissioning:* During this phase the pilot plant results and basic engineering have to be translated into the final plant. The most important consideration here is that the plant should be constructed and commissioned within the time and expenditure envisaged and that it should perform according to the design throughputs and recovery efficiencies assumed in the feasibility study.

Although an activities schedule would indicate various degrees of overlap in real time, these phases will be discrete as they will be defined by policy objectives, a financial budget and a schedule of elapsed time. There will also be discrete subdivisions of activities within each phase.

Therefore, the metallurgist must first distinguish the particular objectives, the kind of recommendation required and the sensitivity of the project, at the particular phase, to his recommendation. The sensitivity will be affected by the extent of the commitment being made and the proportional weight that metallurgical considerations might contribute to the making of the commitment decision.

Thus, from one or a few samples submitted on an ad hoc basis during the prospecting phase, usually the kind of recommendation sought is little more than an opinion or signal to management of the degree of ease or difficulty that might be encountered if an exploration programme succeeded in discovering an orebody similar to the sample. Such an opinion can often be given from the knowledge of the grade and the mineralogy alone with no experimental metallurgy at all. However, the commitment to exploration implies substantial expenditure so that it is important to have a metallurgical interpretation of the mineralogy. If geological indications are strong, metallurgical considerations are unlikely to carry great weight for making a commitment to a preliminary

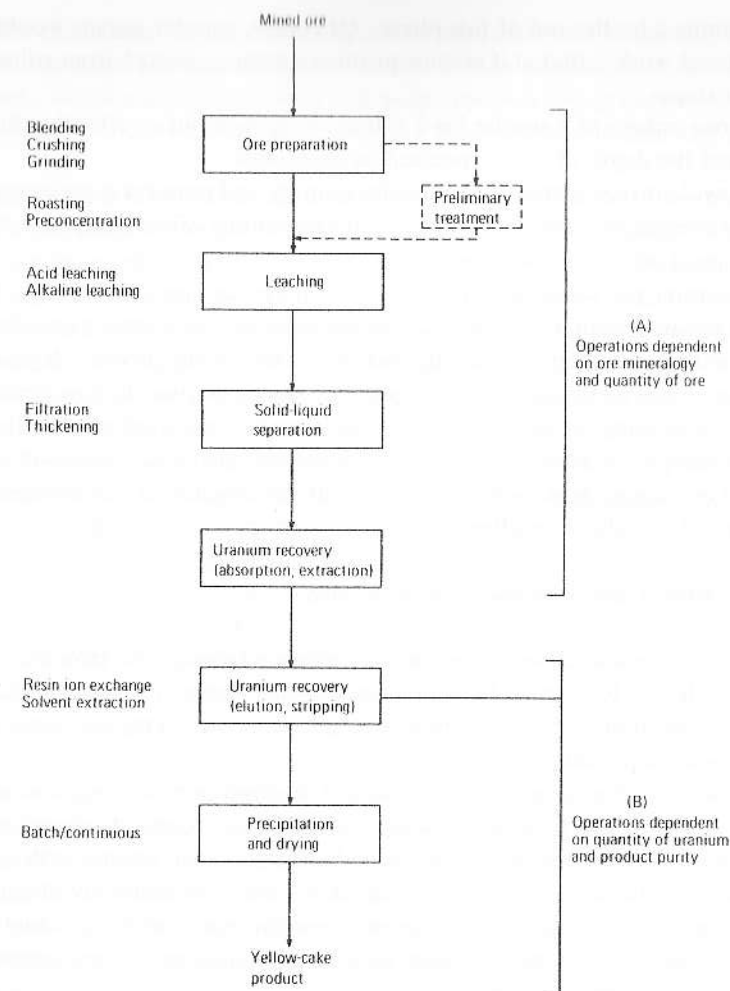


FIG. 4. Schematic flowsheet.

exploration programme (such as preliminary drilling). However, if the mineralogy indicates a rock highly refractory to metallurgical treatment, further commitment should be shifted in emphasis to metallurgical study with only limited field work.

During the exploration stage work will progress to a preliminary feasibility study. However, the metallurgical flowsheet used would be based on implication for many operations. The exception would be the first extraction stages, that is beneficiation (if relevant) and leaching. Much metallurgical study should be devoted to these processes at this phase, so that it is likely that they are close to

being optimized by the end of this phase. Of course, suitable signals would be generated and work initiated if serious problems were suspected from subsequent treatment stages.

Various criteria of a similar kind will apply in each phase, affecting the breadth and the depth of detail required to be studied.

It is evident that a project is multidisciplinary and requires team co-operation with good communications and adequate understanding of each role in order to make the most effective plan of work.

Concerning the communication between mineralogy and metallurgy, the degree of sophistication required in the mineralogical information generally increases with the passage through the various phases of the project. Breadth rather than depth is required at the beginning. It is important to have knowledge of variations existing in the orebody so that basic mineralogical information of a range of samples is imperative. Later on, however, the metallurgist will be interested in learning more about the nature of the uranium in the tailings, particularly if its value is relatively high and stubborn to treatment.

3.3.2. Flowsheet and economic considerations

Section 4 reviews current practice in operating flowsheets. However, at this point it is relevant to discuss the flowsheet in broad terms in order to indicate the relative contribution of the various operations to the overall economy and technology of processing.

A schematic flowsheet of the operations involved in processing uranium ores is shown in Fig.4. The flowsheet can be divided into two parts as shown by the side brackets: that part which is primarily dependent upon the mineralogy of the ore and the quantity of ore to be treated and that part primarily dependent on the quantity of uranium being recovered and the purity of the product required. Grade of ore is the common denominator linking size and mineralogy with specific processing details.

There are some generalizations relevant to the philosophy of planning a programme of metallurgical investigation:

- (a) Many options are available in detail within this broad flowsheet framework and it is seldom that there is only one unique solution to any particular stage. Nevertheless the factor which most influences the final overall pattern of the technology is the mineralogy (and all it implies) of the feed material. Availability of process materials and personal choice are the two next most influential factors.
- (b) The detail of the procedures adopted in the stages following leaching can be much influenced by the beneficiation (if used) and leaching procedures. However:

- (c) If the technology of the treatment can be demonstrated to the end of the leaching stage, the technology is available for the remainder of the flowsheet. Economic problems may still exist (for very low-grade solutions for example) but the technology of recovery from solution is generally assured.

What (b) and (c) in particular imply is that, although detailed work must be done on all parts of the process in order to prepare a specification for the final flowsheet, emphasis and priority should be directed to demonstrating a satisfactory procedure for the "primary" extraction processes, that is the processes involved to the stage where uranium is in solution. These processes include mineral beneficiation if it is relevant, and note that for the sake of completion of this argument it is agreed that the processing could be completed if a suitable concentrate could be produced by physical beneficiation alone. Such has been the case on rare occasions, notably in recovering pitchblende from vein deposits. However, this fact does not alter the basic generality of the above implication.

This implication is supported by consideration of the contribution which the various stages of the flowsheet make to the economics of the overall operation. This matter is dealt with in detail in Section 5. Tables X and XIV illustrate the importance of trying to optimize the operations handling the ore and, within those operations, the leaching process in particular.

It is also implicit in planning technological investigations that undue effort should not be devoted to methods which are patently uneconomic. Good ideas should be tried and the application of excessive quantities of reagents, for example, can be useful in giving basic information, but true innovation requires adherence to economic constraints. Within this project framework a metallurgical programme for a grass-roots project can generally be divided into four phases. The first phase consists of laboratory experiments on preliminary samples to develop a basic, workable treatment scheme. The second phase involves gathering available information concerning the geology of the orebody, the mining plan and the ore mineralogy. The third phase consists of detailed laboratory metallurgical testing on samples representing the range of feeds expected in the plant and the fourth phase, pilot plant test work on one or more samples. The last phase may involve pilot study of the whole process or be limited to certain critical operations.

3.3.3. Samples

The forms that various samples will take have been summarized under 2.4.1. For metallurgical work, samples will generally be confined to outcrop material, percussion drill samples, trench samples, diamond drill core and certain bulk samples from costeens, shafts or tunnels.

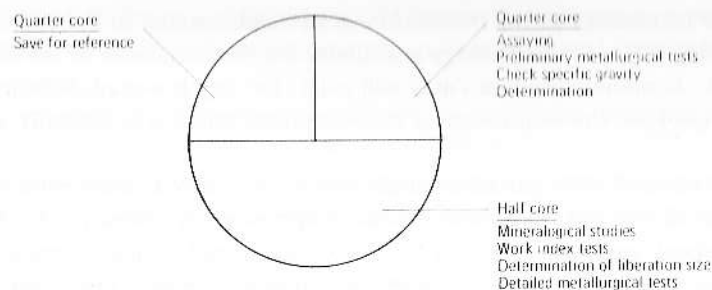


FIG.5. Distribution of drill core (reproduced from Kitagawa [26]).

The outcrop samples will more likely come as the ad hoc samples referred to earlier from the prospecting phase of the project. Experimental work, if required, is confined to a few basic tests of the predictions made from the mineralogy.

Percussion drill samples or pulverized quarter core (from assay sampling) may form the main material available for the preliminary work. However, trench samples or diamond drill core are most desirable to get information concerning comminution and its effect on the processing.

Preliminary work tends to be done on samples as they become available chronologically. However, systematic detailed metallurgical study calls for considerable care in the selection of samples and their blending to form a suitable sample of the whole ore or the specific relevant type of ore to be studied.

The factors and ore characteristics affecting these considerations are: mineralogy, grade, quantity and disposition.

By far the most satisfactory place to resolve the matter of sample selection and blending is in the field, with the whole team involved so that the geological, mining, grade and mineralogical factors as far as known are accounted for.

(a) Distribution of drill core

A diamond drill core costs a great deal and is inherently limited in quantity. Therefore, careful planning is needed to obtain the maximum useful information from this source. The exploration geologist usually arranges for drill cores to be examined initially to log data on the identity of minerals present along with their associations. At this time the metallurgist may also request information on the relative hardness of the ore, the coarseness or fineness of mineral dissemination and its specific gravity. The whole core is used by the mineralogist for these examinations with no loss of material.

The subsequent analyses and testing of drill cores involve assaying, mineralogical studies, grindability or work index tests, mineral liberation size determinations and concentration tests. A suggested method of splitting and distributing core for these purposes is shown in Fig.5.

For those ores that are susceptible to rapid alteration when exposed to the atmosphere, adequate precautions should be taken throughout the test programme to minimize oxidation of the samples. This may include placing the sample in a plastic bag, purging with nitrogen, sealing, and storing in a freezer for subsequent metallurgical testing.

(b) Quantity of sample

The question of how much sample, involves a number of considerations not the least being the methodology of sampling. The previous section discussed the care required in the selection from the orebody of the kind of samples and how they are made up. However, the preparation of a small aliquot of material to be representative of a larger sample of mined ore, or of crushed composite drill core, requires adherence to strict sampling procedures. Here, it is not possible to discuss either the manipulations or the statistics of sampling. These are dealt with in textbooks, handbooks and special publications on metallurgy, mineral dressing and assaying.

However, the mineralogical character of the ore again plays a significant role. Particular care is required, for example, if it is known that the values are distributed with little uniformity or in irregular high-grade spots such as massive pitchblende in vein deposits. Also, a characteristic of many ores (such as the conglomerates and sandstones) is that the uranium is more concentrated in the finer fractions. If a good mechanical sampling system is not available, these characteristics can present formidable problems in reducing coarse bulk samples from development mining to smaller samples representative of the main sample. However, even in sampling crushed drill core they indicate that particular care is needed in order to avoid segregation and loss or disproportionate distribution of fines.

Returning to the question of the quantity of sample for metallurgical study — in early work there is sometimes little choice (as indeed unfortunately there is also little control on the "sampling") and work must proceed on that which is provided. Of course, there is no intrinsic reason why the chemistry should not be the same on a few grams as on large quantities. Work on single small samples (less than 100 g) can supply useful "signal" or survey information during the prospecting and early exploration phase, but should be restricted to that objective. Reports should clearly indicate the nature and limitations imposed by the size of the sample on the subsequent recommendations.

In general, leaching studies should not be done on samples of less than 100 g, but preferably 500 g, which is a good size for routine work, minimizing sampling and material balance reproducibility problems but commensurate with practical containment in laboratory-scale equipment.

To plan a reasonably complete programme for preliminary metallurgical evaluation, approximately 50 kg of sample are required.

Whilst involved in the specific tasks of blending and sampling, there is considerable merit in breaking down the larger 50-kg sample lot (for example, by coning and quartering) to a series of smaller, more manageable lots such as 10 kg. It is then relatively easy to riffle individual experimental lots of 500 g or so from the 10-kg portions.

Similarly, for much larger mine samples (2 to 5 t), it is advantageous to put the material through to the secondary crushing stage and then to divide it by a mechanical device such as a rotary splitter or circularly oscillatory belt distribution, so that the material can be stored in a series of discrete but blended representative lots of say 400 to 500 kg.

3.3.4. The experimental programme

The material presented thus far has indicated that a combination of information on the mineralogy and grade of the ore, together with a working knowledge of the extractive metallurgy of uranium, will allow a considerable narrowing of the initial choice of the line of investigation.

The absence of gangue minerals that are reactive towards acids, e.g. carbonates, would immediately suggest emphasis on an acid leach programme, unless some special constraint – usually not associated with mineralogy alone – suggests that an alkaline (carbonate) leach might be preferred. The identification of refractory uranium minerals will clearly indicate problems ahead and suggest a thorough look at beneficiation procedures. When both sulphides and carbonates are present flotation should be looked at as a means of producing a concentrate for acid leaching and (possibly) a tailing for alkaline leaching.

These decisions are empirical and rely to some extent on the experience of the personnel, both mineralogist and metallurgist. However, this experience is nothing more than an intuitive understanding of the particular ore type under consideration, given a general mineralogical description.

In assessing the amenability of the ore to a leaching process the following parameters must be considered: reagent quantity, reagent concentration, screen size of ore, time, concentration of solids in the slurry, temperature, pressure and agitation conditions.

A few ad hoc experiments are justified with a fixed set of conditions to test the general response of the ore to the type of treatment proposed. However, the most effective way of proceeding is to carry out a set of multivariate experiments, each parameter being varied over a selected range.

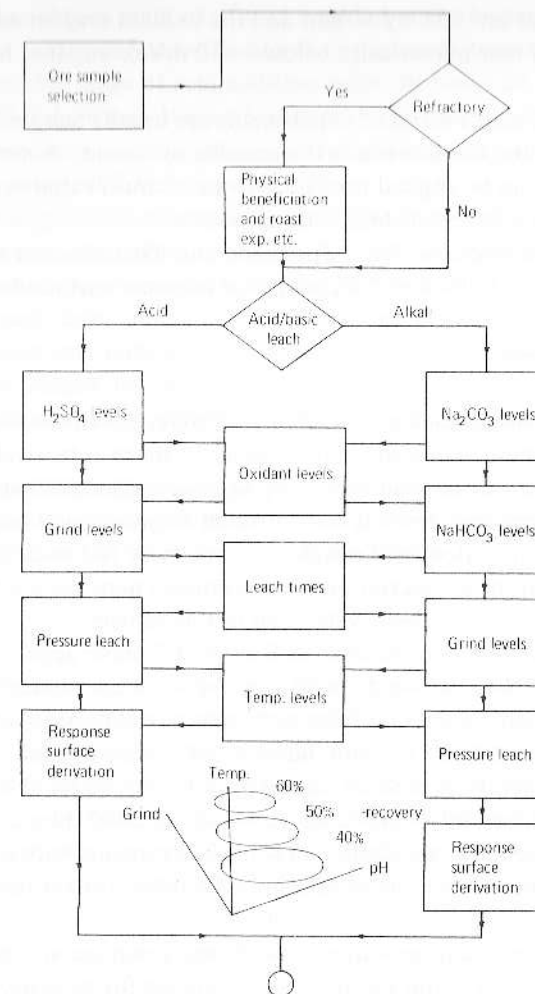


FIG.6. Leaching of uranium ores – a test scheme.

A complete factorial study of all the parameters is seldom justified in the first set of experiments. Such an approach is expensive, difficult to manage and much of the effort rather academic (and there is always the risk of choosing the wrong basic process at this stage). A practical compromise can be made, based on mineralogy and experience with past practice. Some parameters can be fixed in the first series and studied later where relevant. Thus, for acid leaching, ambient temperature and pressure can be used, the agitation can be fixed at a level to produce turbulence through the mixture, the concentration of solids

in the slurry fixed at say 50% by weight and the oxidant reagent added only if the redox potential falls numerically below -450 mV (compared to saturated calomel).

The results of such a series of experiments are usually simple to interpret so far as indicating the trend towards the possible optimum. A number of formal techniques can be applied for the analysis of multi-variate experiments where more complex factorial studies are necessary.

Figure 6 summarizes the above discussion and illustrates as a small insert the application of the "hillclimb" technique of response surface derivation.

(a) Leaching reagents

Refractory minerals such as pyrochlore, uraniferous zircon, thorite, davidite and crystalline brannerite may possibly be more extensively exploited in the future. For these minerals especially aggressive conditions are required and reagents such as nitric acid and hot concentrated sulphuric acid may be required. Relatively exotic innovations will possibly be necessary to handle these materials. With this brief mention the matter must be dismissed here, except to indicate that there is a growing need for a review devoted to this subject.

The data presented and discussed in Section 4.2 make apparent the economic reasons why the alkaline carbonate leach process is not used except under conditions where excessive acid-consuming carbonate minerals are present in the ore. Also, it is unfortunate that a uranium mineral, refractory to acid attack, is usually no less refractory to sodium carbonate solutions so that the alkaline leach offers little hope in that problem. However, unusual things can happen, particularly with metamict minerals, and some experiments with high-temperature carbonate/bicarbonate solutions should be tried if faced with this situation.

Ammonium carbonate appears to have found a permanent place for in-situ leaching where dilute uranium solutions are produced for recovery by ion exchange. This reagent also offers some advantages in agitation leaching, in that it can be recovered by steam stripping if, for example, a resin was used in the pulp process.

Turning to the main process in use, the dilute acid leach, the use of multi-variate experiments as already described is the only way to estimate the correct quantity of reagent (as indeed it is for the alkaline leach as well). The acid consumption is mainly controlled by mineralogy but the final incremental variations are strongly dependent on temperature, time and the actual concentration of the acid, as these parameters control the rate of attack on the less reactive silicate minerals in particular. To study the effect of the acid concentration itself, it is advantageous to plan two series of experiments in the preliminary study.

In one series a chosen amount of acid is added at the beginning of the leach and the free acidity measured during the progress of time but no further acid added. A suitable range of acid addition might be based on the amount of ore such as 20, 30, 50, 80 and 100 kg/t. Alternatively, the quantity of acid might be such as to have, at the commencement of the leach, a certain concentration of acid such as 0.1, 0.2, 0.3, 0.4 and 0.5N.

In the second series the experiments should be done under conditions of controlled pH, using a range such as pH 1.0, 1.5, 2.0, 2.5, acid being added during the experiment to maintain the chosen pH.

Two-stage leach techniques should be examined if these experiments indicate that a relatively high free acidity is required to attack the uranium but, owing to lack of reactive gangue, much of the acid remains at the end of the leach. This is similar to the situation with the brannerite in conglomerates at Blind River.

Of course, oxidation conditions must be attended to, usually by adding sufficient oxidant to maintain the redox potential at -450 mV, or numerically greater (with respect to the saturated calomel electrode).

(b) Temperature

Increasing the temperature will cause an increase in the rate of leaching the uranium. The use of temperatures between 65 and 75°C is common practice in the Witwatersrand and Elliot Lake areas in order to achieve satisfactory uranium dissolution. However, for many ores the increase in temperature will also increase the rate of attack on many gangue minerals with consequent increase in acid consumption and other possible deleterious effects on the composition of the liquor. These effects can only be determined by experiment. However, in considering the flowsheet conditions, the effect of the process conditions themselves, including the ambient temperature of air and water supply, must not be overlooked.

Grinding heats the slurry and the heat of dilution of sulphuric acid is significant. These effects can be relatively accurately estimated by calculation and, if for no other reason than to correctly simulate the future operation, laboratory experiments may require some small heat input. In at least one operation (Mary Kathleen) a laboratory study indicated a distinct advantage to be gained by controlling the temperature rise caused by acid dilution and led to dilution and cooling prior to addition to the leach.

(c) Oxidants

Sodium chlorate and peroxides are convenient oxidants to handle (but present a hazard which must not be forgotten!). Sodium chlorate can lead to

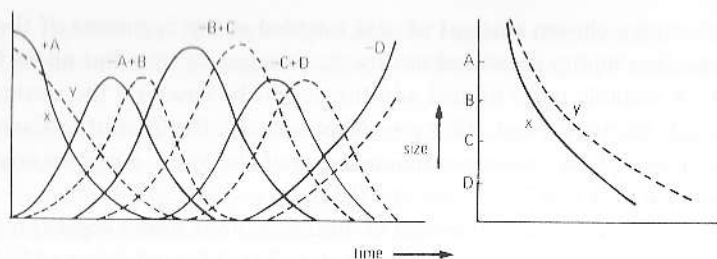


FIG. 7. Schematic diagram showing relative rates of comminution of two minerals, x , y .

an accumulation of chloride ions in the barren tailings and therefore limit the ability to recycle.

Pyrolusite (manganese dioxide) need not be very high grade and if there is a source within reasonable distance it can often be mined and milled for supply more cheaply than the above oxidants. An advantage of manganese dioxide is that it reacts to convert Fe^{2+} to Fe^{3+} at lower temperature and acid concentration that does sodium chlorate.

As far as preliminary experimental work is concerned any of the above oxidants can be used. The final choice of oxidant is almost entirely a matter of economics, unless the use of oxygen itself is contemplated when more sophisticated pressure technology is involved.

(d) Comminution and screening

The amount of study to be devoted to this parameter and operation depends upon the degree to which physical beneficiation is being considered in preference to direct leaching. In two commercial techniques at present being practised comminution is either unnecessary (in-situ leaching of sandstones) or only applied for primary breakage (dump leaching).

For agitation leaching, comminution is a necessary operation. However, in acid leaching it is not unusual to find that the final size of grind specified is selected for convenience of handling the slurry. This is the case in many operations processing sandstone ores, where even complete release of the individual sand grains is not always necessary for leaching. Dense ores require finer grinding to expose the uranium mineral and relatively fine grinding is required for alkaline leaching.

It is economically desirable to avoid overgrinding. However, in contrast to the requirements of physical beneficiation, the production of "fines" in itself is not deleterious in the leaching operation.

One disadvantage usually claimed from overgrinding is a deleterious effect on the subsequent solid liquid separation processes. Although this fact is not disputed, the degree of this problem is more a property of the mineralogy such as the presence of inherently soft or fine minerals — clays, micas and chlorite.

However, if physical beneficiation is being considered, comminution is more than an operation of simple size reduction. It becomes one of the important integral parts of the process. Physical separation processes become increasingly difficult as the size of the whole ore becomes finer and there is no physical process which is satisfactory for separation of minerals in the extra fine sizes (say less than $5 \mu\text{m}$).

In addition, although all minerals in an ore undergo a reduction in size during comminution, the rate at which each appears in the finer fractions varies depending upon the original particle size and the rate at which it grinds. Figure 7 illustrates this effect in the form of a sizing distribution of the mineral components.

No matter what process might be favoured in the initial planning the inclusion of work on the sizing distribution is important. Apart from giving a lead to the possibility of beneficiation, the information forms a valuable background to the whole treatment problem.

Some of the techniques used by the mineralogist for analyses of sizing distribution have been discussed in Section 3.

Work Index estimation: The determination of Work Index using the Bond grindability method is necessary at some stage to assess the power and the size of the mill or mills required.

The Bond test is performed in a 305-mm-dia. ball-mill (dry) and a relatively small quantity of ore is required. This makes it possible to test drill core samples of deep-level ore. For maximum information work index tests should be carried out in conjunction with the tests to determine the mineral liberation size. Rod-mill work indices can be determined for any mesh from 4 to 65; however, the normal range is from 8 to 28 mesh. The ball-mill work index can be determined for any sieve size below 28 mesh. It is important that the mesh of closing sieve selected for both tests is such that the ground product from the test is slightly finer than the product required in the commercial plant. Failure to take this precaution may result in undersizing the grinding units for the commercial plant, especially with ball mills, as the power requirement to grind minerals that are already at their natural grain size may increase very sharply with any increase in size reduction.

If autogenous or semi-autogenous grinding are to be considered for the commercial plant, the appropriate grinding tests should be conducted. A simple method of predicting autogenous grinding mill requirements for processing ore

from a new deposit has recently been published by Loveday (1978 [34]). A method has been developed for testing semi-autogenous milling at a small scale (McPherson 1977 [35]) and the results of this test could be used in a preliminary feasibility study. However, pilot-scale tests will be necessary to determine fully-autogenous and semi-autogenous milling characteristics for the plant design. The use of this type of milling eliminates secondary and tertiary crushing and can result in savings in capital and operating costs. However, relatively wide changes in product size and plant capacity can occur as a result of changes in feed quality.

Comparative work indices are also sometimes useful in deciding on the alternatives of finer grinding or the use of sand/slimes classification. Thus sand-stones and granites can show a steep increase in grindability when the "natural" grain size of the quartz particles is reached or, to be more specific, most of the work is transferred from breaking up the rock into its "natural" grain size to reducing the size of those separate mineral grains.

Advantage can also be taken of mill vendors' facilities in testing the applicability of their products to grinding in particular.

(e) Physical beneficiation

As referred to in the introduction to this Section, with a few notable exceptions physical beneficiation has not been widely used in processing uranium ores. However, the potential economic advantage, if it can be used, is substantial and this increases as the grade of the ore goes down, so that no investigation should proceed without some consideration of the potential of the various methods available.

The earlier in the processing that beneficiation procedures can be used the greater the potential saving. For this reason selective mining techniques and radiometric scanning of ore trucks are sometimes used and can be classed as important beneficiation procedures before the ore enters the battery limits of the mill.

There are several useful techniques which can be applied to coarse rock.

Heavy medium concentration may be used if the host rock is mixed with country rock of different density. This technique was very successfully applied at the Radium Hill mine to treat the ore after the secondary crushers, the feed size being minus 25 mm plus 1.67 mm.

The density of various rock types can be determined using an air pycnometer and this information correlated with grade. A stepwise change from low to acceptable grade which coincides with a reasonable incremental change in SG is a good indication. In modern equipment separations can be made with an SG difference of 0.1. Finer sizes (down to 150 μ m) can be treated in the cyclone which will make a reasonably clean separation with a difference of 0.01 to 0.02. However,

TABLE III. ASSAY VALUE OF TOTAL 3-in. SAMPLE 0.258% U_3O_8

Reproduced from Harris and Steele (1960 [36])

Assay range of U_3O_8	% Weight of sample	Average assay % U_3O_8	Distribution of U_3O_8 (total)
Less than 0.02	24.7	0.008	Rejected by sorter as waste
0.02 to 0.04	8.2	0.025	
0.04 to 0.05	nil	—	
0.05 to 0.06	nil	—	
0.06 to 0.07	1.0	0.068	
0.07 to 0.08	nil	—	Cut-off set 0.07%
0.08 to 0.10	nil	—	
0.10 to 0.20	2.0	0.110	
0.20 to 0.40	36.5	0.270	
0.40 to 0.75	27.6	0.570	
			Reported on ore belt

good desliming is mandatory for both pool and cyclone separators. Therefore, a significant weight fraction of the ore must be liberated in the coarse fractions to make the operation economically meaningful.

Radiometric and colour sorting are also techniques to be considered for beneficiating before grinding. Once again the favourable indication is to demonstrate a significant incremental change in grade (for radiometric sorting) or in grade and "colour" (for colour sorting) from waste to ore rock. Radiometric sorting has been used in Canada and Australia.

Table III gives results of the preliminary study carried out to test the potential of sorting for Mary Kathleen and indicates the favourable "break" in grade.

Considering techniques applicable to ground ore a simple form of beneficiation is screening or similar size classification, and a size distribution analysis will immediately indicate the potential of this technique. Consideration can also be given to sending a coarse grit and sand fraction of low grade to a heap leaching operation. If, following a basic sizing distribution analysis from a mineralogist, a more extensive investigation is undertaken into a range of beneficiation processes, it is useful to commence with a more detailed analysis, applying simple separation techniques in the laboratory to provide products for the purpose of the analysis.

If the mineralogical analysis has shown that the uranium minerals are coarse grained, gravity concentration using jigs, cones, spirals or sluices may be possible.

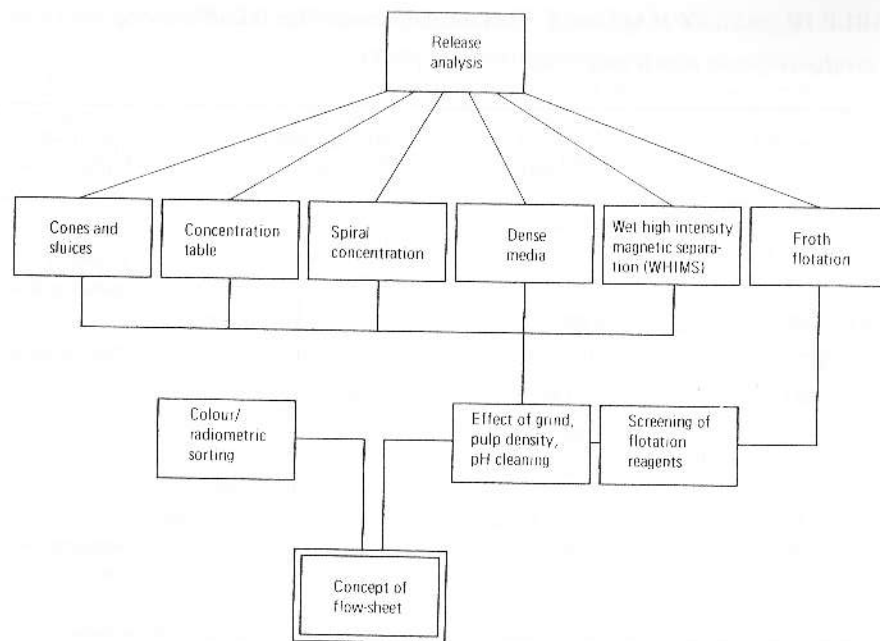


FIG.8. Screening of preconcentration processes. (WHIMS = Wet High Intensity Magnetic Separation.)

The techniques described in Section 2 for heavy liquid separation analysis of size fractions will give a good indication of the potential of the techniques. For example, if separation is not effective in heavy liquid at SG 3.3 it is unlikely to be effective in most mechanical gravity devices. In addition, most gravity machines are also sizing machines. The best combination is to have the heavy mineral fraction in the fine end of the size range being considered, always bearing in mind, however, that the very fine "slime" fraction cannot be treated. A test on a laboratory shaking table will indicate the best results that are possible using these devices. The ore should be ground to pass 300 μm and a portion of the ground ore should be split into size fractions by hydraulic classification. The classified fractions will give the best results on the shaking table (and on the other devices), but the circuit becomes more complex. A few tests of this type will give a good indication as to whether gravity concentration is worth pursuing.

Magnetic concentration (high and low intensity) should be considered. Gravity concentrates and a suitably ground sample of the ore should be subjected to wet high-intensity concentration at progressively higher field strengths. As with gravity concentration, the efficiency of wet high-intensity magnetic separation

decreases as the particle size becomes smaller. Therefore, in both cases it may be necessary to deslime the feed and only upgrade the sand fraction. An analysis of recovery as a function of particle size will indicate the size at which upgrading becomes economic.

Finally, flotation should be considered as a means of upgrading uranium ore. Flotation has been used to concentrate davidite and brannerite and separate them from reactive silicates so that strong acid attack could be applied. It has also proved useful in flotation of pyrite fractions from carbonate in the USA and from gold tailings in South Africa. Fatty acid collectors with or without combination of fuel oil has been used for the oxide minerals and xanthates for the sulphide flotation. Indications that flotation might be applicable can only be examined by empirical evaluation of a range of reagents. Water quality can be important in fatty acid flotation particularly.

Of course, any indication of success in beneficiation must be followed by study using laboratory or pilot-scale machines simulating the prototype operation.

The results of all these upgrading tests may be determined by monitoring radiation with a rate meter, which will indicate the degree of concentration, and a few samples of the ore, concentrate and tailings should be assayed for calibration purposes. Figure 8 summarizes the screening process which might be applied for initial evaluation of beneficiation.

(f) Roasting

Roasting of the ore before leaching can be applied for a variety of reasons such as the following:

- Modification of gangue minerals such as montmorillonite, to enable the solid/liquid separation to function after leaching. It is believed that this is being investigated for the Yeelirrie calcrete ore for alkaline leaching where clay minerals have created severe problems.
- Modification of the gangue to release uranium and also possibly reduce the weight of ore to be leached. This would apply particularly to carbonaceous shales etc. This is one case where extra fuel energy may not be required.
- Recovery of vanadium — salt roasting is a classical technique although strong acid leaching is now generally preferred.
- Attack of refractory mineral — fairly drastic but may be the only way.

It is not uncommon to find that the solubility of the uranium will either remain satisfactory or improve over a given temperature range and then deteriorate quite rapidly with further temperature rise as more stable compounds are formed.

Differential solution can sometimes be effected by taking advantage of the different thermal stabilities of metal sulphates. Experimental work on roasting is aimed at delineating these factors and the application of the thermal balance and of differential thermal analysis and mineralogical techniques can be very informative.

(g) Alternative leaching procedures

Alternative methods to agitation leaching for extracting uranium into solution include bacterial leaching, heap leaching, in-situ leaching.

In fact, bacterial-assisted leaching takes place to some extent in both heap and in-situ leaching. However, the main indication for a particular study of this technique is the presence of pyrite or of a nearby supply of pyrite, particularly for application to heap leaching of low-grade material. Encouraging results have been obtained in establishing the bacterial process in a separate "fermenter" to produce ferric ions and acid for circulation to the leaching site, thus alleviating the problem of maintaining oxygen supply to the heap.

Laboratory investigations into any of these techniques are usually confined to establishing an initial formula for the leach solution and to studying the basic chemistry of processes which might interfere or enhance the operation. It is apparent that if leaching does not proceed under agitated conditions in a laboratory there is little point in applying the conditions in heaps or in situ. However, having established such a basic formula, further experimental work in the laboratory will soon enter a phase of rapidly diminishing returns, so far as scale-up interpretation is concerned and work will be required on heaps or in bore holes. Obviously, both geological and mineralogical indications must be favourable for in-situ leaching to be considered. Thus, lack of porosity, or irregular "honeycomb" porosity, steeply dipping porous beds and lack of impervious containment strata are some contra-indicating factors. Carbonates are contra-indications for acid leaching and sulphides (which consume soluble oxidants rather avidly) and to some extent gypsum contra-indicates for alkaline (ammonium carbonate) leaching.

(h) Solid-liquid separation

Mineralogy has a considerable influence on the dewatering processes of thickening and filtration and their very important function of washing. The introduction in the late 1950s, and subsequent development of synthetic polyacrylamide flocculants, has relieved this problem, particularly in the acid circuits. These fast acting flocculants, however, do not always succeed in producing a clear supernate and it is advisable to study the effect of combining some of the slower acting (and cheaper) natural product flocculants such as the guar gums with the polyacrylamides, mixtures which can sometimes achieve the high settling rates with good clarity at a lower cost.

Early in the programme there will usually be an indication from the mineralogical report as to whether this operation will be a problem. However, settling tests in cylinders and filtration measurements, using small filter leaves, should be included in any work in which any significant change has been introduced such as in the ore type (sample) or leaching conditions.

Textbooks and handbooks give adequate background information. Good empirical procedures have been developed by filter manufacturers (who will also often supply the laboratory filter leaf) which can be interpreted for scale-up with reasonable accuracy.

For the evaluation of thickener size, the method of Couche and Goldney (1959 [37]), an empirical adoption of the Talmage and Fitch analysis, can be recommended. It has been confirmed in pilot plant studies and used in subsequent design of several operating counter-current decantation (CCD) thickener circuits.

(j) Other stages in the flowsheet

Mineralogy affects all the stages in the process, in the sense that it is reflected in the leach liquor composition, which in turn is the main reason why the specific details of the flowsheet of one plant vary from that of another. There are no fixed rules for indicating exactly how a particular combination of ions will behave in the extraction process, or to what degree it will affect the purity of the product. All the high valency metals, elements forming high valency anions or complex anions, are suspect and components such as silica and titania, which tend to undergo irreversible hydrolysis and polymerization, are possible causes of problems. The presence of metals such as molybdenum and cobalt, to which the specification of the product is particularly sensitive, may indicate the need to introduce a process variation.

In the preliminary programme, work can be confined to carrying out resin equilibrium and solvent shake-out experiments to plot equilibrium curves and see that they conform to expectation.

Spectrographic analyses of selected feed and product liquors at that stage will indicate the presence of problem elements. However, the degree to which problems might arise from irreversible reactions causing accumulation of reagent "poisons", and polymerization leading to resin blockage or solvent interface "crud", can only be studied in a multi-cycle continuous system, simulating operating practice. This work should form part of the later detailed study.

3.4. Preliminary assessment

At the completion of the exploration phase of a project, the metallurgical study should be completed to the end of the preliminary experimental investi-

gation and adequate background information gathered by the metallurgist for him to recommend the appropriate basic flowsheet and preliminary estimates.

A flowsheet showing mass and volume balances should be constructed and written down as early in the programme as possible and as soon as a process is defined, even if much remains conceptual and requires experimental verification. Such an exercise will indicate areas in the process to which the economics are most sensitive to uncertainties and put in perspective the magnitude of the whole process as well as its various parts.

Absolute amounts of recoveries and losses can be assessed. In comparing the technical results of two or more processes or conditions of processing, the percentage recovery is commonly used as it is a useful index of comparison. However, percentage means very little when comparing the economics of alternatives except to indicate that there is more or less material yet available. The actual amounts and value of uranium won or lost are the numbers that count and the mass and value of materials determine the magnitude of equipment required.

In addition, such a flowsheet identifies side-streams and recycle-streams that must be accommodated in the process. Thus, backwash solution from sand clarifiers does not disappear. If discarded it is a waste of water and will carry some uranium to waste. If it is recycled to the wash circuit (the usual procedure) that circuit may not be quite as efficient. Likewise, solution from special scrubbing or wash elution stages in SX and IX must be accounted for.

Therefore, writing down such a balanced flowsheet early in the investigation can be very informative. The background information required to supplement the results of this metallurgical work is the input from the other disciplines in the project team and covers the following forms:

Geology and mineralogy. The general geology, geological ore reserves and the disposition of the mineralization within the deposit, in other words the detailed mineralization model. There should be a clear understanding of the drilling plan and trenches or shafts developed to produce the samples and, of course, the variation of mineralogy and grade.

Mining. The basic concept proposed for mining the material should be known along with initial ideas on cut-off grades and possible dilution; also the type of diluent rock which might be expected.

Services. It is necessary to know the availability of essential services — the source, quantity and quality of the water supply; availability of power; and the quality of access for the transport of equipment during construction, and daily to and fro of goods and services during operations.

Site location. Much of the above information will be considered, together with other geographical and sociological constraints, to develop an early concept

of the site, its development and the constraints it may place on any flowsheet considerations.

Costs and supplies. Finally, it is important to have developed basic data for estimating. Questions might be asked such as those concerned with unit costs of earth moving, foundations and similar civil work in the area; costs of steel on site; any special award loadings. Enquiries should be made concerning the availability and supply position of steel and similar basic materials, and any especially large equipment such as crushers, mills, large motors and electrical switch gear.

Input to project. At this stage the metallurgist should be expected to develop data to compare alternatives, particularly developing comparative treatment costs for various grades and mill capacities. Such data must be available to combine with alternatives developed for mining, in order to generate and optimize an overall mining plan and project model. Every block of rock must be given a value or range of values to estimate whether it is ore or waste and so develop the total concept.

Although much detailed metallurgical study can be proceeding in the development phase of the project, it is only after the completion of the above considerations that the essential parameters affecting the final flowsheet are fixed and work can proceed to the detailing of this flowsheet specification.

3.5. Pilot plant

Following the determination of the preliminary process flowsheet, the development of that flowsheet involves study and evaluation of detail. Much of this work has to do with the operation of the process and the hardware to carry out the operations. Materials handling characteristics, for example, must be considered. This is often left to a contractor to cope with, but unfortunately all too often with sketchy specifications, under the mistaken impression that one ore handles like another. The consideration of detailed conditions in such processes as solvent extraction, ion exchange and product precipitation all come into this stage.

Although highly relevant to the broad subject under discussion, it is beyond the scope of this report to deal with these aspects in detail. It is plainly necessary to develop the subject of metallurgical evaluation of uranium deposits. However, the pilot plant and its role require some mention before closing this section.

First, the idea that as night follows day so a pilot plant follows laboratory work, is not valid. Some work will usually be justified using pilot-scale equipment and major innovations affecting hardware or unit operation design parameters require detailed pilot-scale study. However, a total flowsheet operation on a continuous pilot scale is seldom required for process and engineering design purposes

alone (training objectives are dealt with below). The continuous operating pilot plant is certainly not a satisfactory medium for experimental work, which can rarely be adequately controlled. Experimental work is best done in the laboratory or by isolating a particular operation for study in pilot-scale equipment, under batch or simulated continuous operation. The continuous pilot operation is the place to demonstrate that all operations can be run continuously together at a nominated daily rate, to test the effectiveness of the chosen conditions, measure data to evaluate the effect of recycle streams, and detect any long-term effects of trace components and get some measure of the performance of materials. Indications of problems with materials handling can be obtained (but not necessarily cured at this scale of operations). Important detailed "tuning" to the flowsheet can be done, but if any serious problem arises it is best to "go back to the drawing board", in this case the laboratory or isolated batch study for experimental work.

The objectives of proceeding to a continuous pilot-scale operation must therefore be the subject of very deliberate definition. Frequently the purpose has more to do with the education and training function, which is a very valid objective and thoroughly justifies such an operation in relevant circumstances. However, if these are exposed as key objectives, the operational plan must reflect them. Exercises in developing operating procedures, sampling schedules and operator training, for example, can be most valuable experience before having to do the same tasks for the future large-scale plant.

4. INDUSTRIAL ORE-PROCESSING PRACTICE

4.1. Uranium ore-processing technology

An outline was given in Section 3 of the various unit operations involved in processing uranium ores. As shown in Fig.5 the overall circuit can be divided into two major parts. The first part is dependent primarily upon the mineralogy of the ore and the quantity of ore to be treated; the second is principally influenced by the quantity of uranium being processed and the required product purity.

Nearly all the current industrial operations are based on two general flowsheet arrangements. These two basic flowsheets are illustrated in Figs 9 and 10. Figure 9 shows the unit operations that are used in the acid circuit which is used by most plants.

The ore preparation, grinding, and leaching operations of most acid leach circuits are similar, but after the leaching step two different routes have been followed. In one type of operation (A), a solids-liquid separation is made on the leach product slurry, and either resin ion exchange or solvent extraction is used to recover the uranium from the clarified liquors. In the second general type of

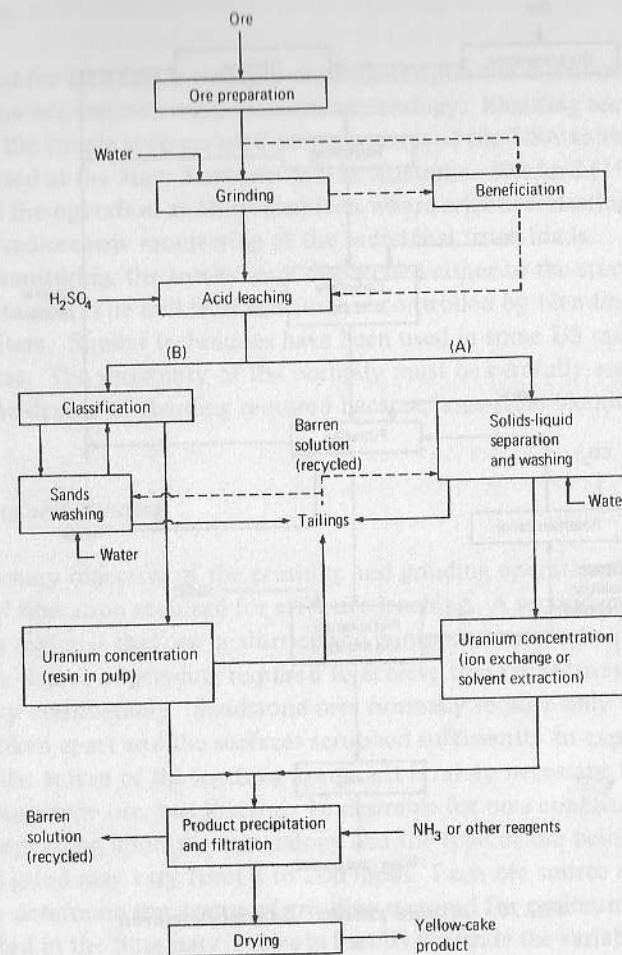


FIG.9. Acid process: General flowsheet.

operation (B), the leach is separated into sand and slime fractions. The sand fraction is washed, the wash solutions are combined with the slimes and the soluble uranium is recovered directly from the slimes slurry by resin-in-pulp (RIP) techniques. After this point, the remaining operations are again similar.

Figure 10 illustrates the steps in the alkaline circuit, which was developed to treat limestone ores and other materials in which the carbonate content was too high for economic processing with acids. Most of the alkaline leach operations have used flowsheets similar to that shown in Fig.10.

These two flowsheets are basic to industry, but a wide variety of equipment arrangements have been used to meet the specific needs of individual milling

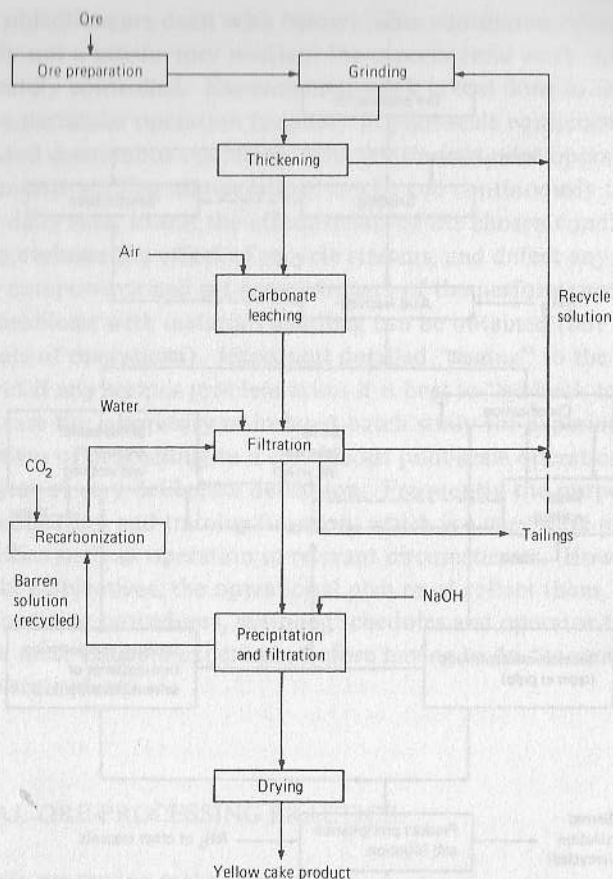


FIG.10. Alkaline process: General flowsheet.

operations. The following sections discuss the various unit operations and equipment used for these flowsheets, and also examine the general influence of ore mineralogy on the unit operations.

4.1.1. Ore preparation

Ore preparation includes operations such as blending, crushing, grinding, roasting, and beneficiation. The objective of these operations is to prepare a feed that will permit the best balance of uranium recovery and minimal costs in the subsequent processing operations. Some ores may require all of these preparation operations, others may need only one or two.

(a) Blending

The need for blending is strongly dependent upon the distribution of uranium in the ore and on variations in its mineralogy. Blending techniques have ranged from the simple systems used in many parts of the USA to the complex procedures used at the Mary Kathleen mill in Australia. Wreford (1965 [38]) has reviewed the operation at Mary Kathleen where selective blasting was used followed by radiometric monitoring of the individual truck loads.

After monitoring, the trucks were dispatched either to the stockpile system or to waste dumps. The mill feed was further controlled by blending from the stockpile system. Similar techniques have been used in some US mills processing sandstone ores. The variability of the orebody must be carefully assessed to determine the degree of blending required because large-scale blending systems are costly.

(b) Crushing and grinding

The primary objective of the crushing and grinding operations is to produce the degree of liberation required for effective leaching. A secondary objective is to produce a material that can be slurried and pumped through the processing circuits. The degree of grinding required to achieve these objectives for different ores may vary considerably. Sandstone ores normally require only that the sand grains be broken apart and the surfaces scrubbed sufficiently to expose the minerals to the action of the leaching agents. It is rarely necessary to break the grains of a sandstone ore, but this may be desirable for ores containing refractory minerals. Depending upon the mineralogy and the type of ore being processed, the required grind may vary from 3 to 200 mesh. Each ore source must be evaluated to determine the degree of grinding required for optimum operation. The data listed in the Summary Tables in Part III illustrate the variability in grinding requirements, but some general trends are discernible. For example, the grind for most acid leach operations on sandstone-type ores has been in the minus 28-mesh to minus 35-mesh range, while that for limestone ores treated by alkaline leaching has normally been minus 65 mesh. The grinding required for conglomerate ores is more variable and is often expressed as the percentage of material passing through a 200-mesh screen. A grind of 50% passing 200 mesh is not uncommon, but for some ores much finer grinding may be desirable. It has been possible to leach some ores such as the granite-type Rossing deposit, at much coarser sizes.

Many uranium plants have used conventional three-stage crushing followed by open-circuit rod mills and closed-circuit ball-mill systems. In some sandstone operations, the ball mills have not been necessary. A relatively recent trend in mills processing sandstone ores is to accomplish both the crushing and grinding in semi-autogenous mills. In these operations the run of mine ore is fed directly to an

autogenous mill that is operated in closed circuit with a sizing device such as a DSM screen. If applicable, this type of comminution system can simplify operations and decrease costs. This type of autogenous or semi-autogenous cascade mill has been installed in several US uranium mills in Wyoming. Semi-autogenous wet grinding in pebble mills has also been practiced on conglomerate ores in South Africa and Canada.

Metallurgical mineralogy can be an effective guide both for selecting the type and degree of grinding that should be investigated for any new ore, and also for trouble-shooting the grinding when difficulties develop in the grinding circuit of an operating mill.

(c) Beneficiation and pretreatment

As discussed in Section 3, the mineralogy of uranium ores has limited the use of physical beneficiation in uranium ore processing. The composition and characteristics of most uranium ores are such that it is often possible to upgrade a given ore, but the uranium recoveries are usually not adequate, and the uranium content of the tailings are unacceptably high. However, consideration should always be given to the potential for physical beneficiation because a number of economic advantages are possible. These advantages can include the following:

Transportation costs from a remote mining location to a central processing plant could be reduced significantly;

The capital costs for the ore treatment section of the processing plant (Section A of Fig. 5) may be appreciably lower. This effect becomes increasingly important as the grade of the ore decreases;

Removal of reagent-consuming gangue materials can reduce the leaching reagent requirements and other process interferences.

Physical beneficiation may also offer the opportunity to concentrate the more refractory uranium minerals into a stream that can receive special, more severe treatment. This type of procedure may have an advantage for ores that contain refractory minerals such as davidite or carbonaceous uranium-bearing components.

As previously described, the techniques of metallurgical mineralogy can provide an effective guide for determining the beneficiation potential of a given ore.

A wide variety of physical beneficiation or ore pretreatment techniques have been investigated, some of which have been applied on an industrial scale. The techniques include the following:

Radiometric sorting: In this technique the emission of gamma radiation from the ore is utilized to reject low-grade waste. It is most successful in industrial

practice where there is a stepwise variation in uranium grades. It has been used primarily to remove gangue materials prior to the final crushing and grinding operations. The operating principle of the radiometric sorter is to pass successive pieces of rock past a radiation-sensing element which counts the total disintegrations rate per unit for each piece. If below a preset figure, an electronic signal actuates a mechanism which rejects the piece. The system is limited to use with coarse-sized rock from 2.5 to 30 cm. Only one mine, located in the United States of America, continues to upgrade uranium ores by radiometric sorting. Photometric and conductometric sorting of uranium ores is in the early development stage. Harris and Steele (1960 [36]) have described an application in detail.

Gravity concentration: A gravity concentration flowsheet using Reichert cones, spirals and tables is used by the Palabora Mining Co. [28] to produce a concentrate of heavy minerals. Magnetic separation is also included in the flowsheet to reject magnetite and flotation to scavenge copper. The final uranium-bearing concentrate is primarily a combination of magnetite and uranium minerals.

Heavy-media separation: Heavy-media cyclones have been used in South Africa to treat the conglomerate gold-uranium ores. At the Vaal Reefs Gold mine a heavy-media cyclone system was used to produce a low-uranium overflow that was bypassed directly to the gold-leaching operations. The uranium-bearing underflow was fed to the acid-leaching circuit. A combination of heavy media with flotation was used to produce davidite concentrates at the Radium Hill operations in South Australia [27]. In this operation an ore containing about 0.2% U_3O_8 was concentrated to 1% U_3O_8 with an 86.4% recovery. The ore could not have been treated without this prior concentration step because the davidite mineralization required severe processing procedures that would not have been economic on the total ore.

Flotation: Many research organizations have investigated the froth flotation of uranium ores, but a relatively small amount of the work has been reported because it has seldom been possible to produce a discardable tail. Flotation, however, has been successfully applied in a number of operations where partial concentration or separation of the uranium values was desirable. Some applications of flotation are listed below:

Concentration of uranium, gold, and pyrite from the Witwatersrand ores in South Africa;

Separation of copper concentrates from vein, limestone, and sandstone ores in both Canada and the United States (Beaverlodge, Saskatchewan), (Moab and Mexican Hat, Utah). The primary objective of the US practice was to produce copper-uranium concentrates for separate treatment. This procedure reduced both tailings losses and process interferences in the main plant circuits.

Production of low-lime concentrates for acid leaching at the Beaverlodge operations in Canada. Considerable experimental work on the separation of carbonates from US sandstone ores has also been conducted, but no commercial circuits have been installed. Apparently the uranium losses to the carbonate product have been unacceptable, and separate treatment circuits for this product have not been considered economic.

Magnetic separation: Magnetic separation has been included in a few flow-sheets such as the Palabora mill and the Porter Brothers operation in Idaho. High-intensity wet magnetic separation is being investigated in South Africa, but no industrial applications are known.

Sizing separations: Upgrading operations based primarily on sizing have been conducted on sandstone ores in the USA. A primary objective of these applications was to reduce haulage costs from distant mining locations. When most US sandstone ores are ground to about minus 28 mesh, nearly 80% of the uranium is found in the minus 200-mesh fraction, which constitutes about 20% of the total ore weight. If the plus 200-mesh fraction is leached, and the uranium is then precipitated and combined with the minus 200-mesh fraction, overall recoveries in the middle 90% range are possible. This type of procedure was used by the Union Carbide Corp. in its Rifle Complex which included operation at Rifle, Colo., Slick Rock, Colo., and Green River, Utah.

The foregoing examples are not a comprehensive listing of physical beneficiation applications, but are intended to illustrate the types of operation that have been used. Experience has shown that the physical beneficiation potential for each uranium deposit may be specific, and must be individually evaluated. Good metallurgical mineralogy is a key to assessing this individual potential.

4.1.2. Leaching

The leaching characteristics of an ore are determined by the mineralogical composition, but broad generalization only is possible, and experimental studies are required to determine the most applicable leaching conditions.

The general chemistry of uranium leaching and the methodology for experimental studies have been covered in Section 3. In the following paragraphs emphasis is placed on the type of equipment that has been used for uranium leaching operations and on the circuit arrangements and techniques that have been employed.

(a) Leaching techniques and equipment

Nearly every type of agitation leaching equipment has been used in uranium leaching circuits. In the early mills, the choice often was based primarily upon the

past experience of the operating company. More recently, design criteria have also included experimental evaluations of the agitation requirements. A series of general comments on the various types of in-plant leaching equipment is presented in the following paragraphs:

Mechanical agitation: Mechanical agitation with propeller or turbine-type agitators is probably the most widely used mixing technique. In recent years, the adoption of large, high displacement, axial flow propellers has significantly reduced the horsepower requirements for many operations. The abrasion characteristics of a given ore pulp are determined by shape of the ore particles and the mineralogical composition. For example, quartz particles larger than about 35 mesh can be very abrasive, but if significant amounts of clay minerals are present the effect can be sharply diminished.

Pachuca leaching: Pachuca leaching units have been used successfully in acid leach operations in South Africa, Canada and other countries. This type of leaching unit has also been favoured for alkaline-leaching operations in the USA. The mixing action in a pachuca is relatively gentle, and if a scrubbing action is desirable during leaching, mechanical agitation may be preferable. A mild scrubbing or attritioning can increase the leaching rate for sandstone-type ores.

Autoclave leaching: Autoclave leaching has been used in many alkaline leaching operations. Autoclave units permit higher operating temperatures which significantly increase the leaching rates. Because the operating pressures for most alkaline leach autoclaves are usually less than $6.9 \times 10^5 \text{ Pa}^1$, feed-slurry pumping has not been a major problem; however, the abrasion characteristics of the pulp should be considered carefully when the pressure let-down system is being designed. Autoclave systems for acidic leaching are reported to be in operation in the USSR, but have not been applied elsewhere. The use of autoclaves for acidic leaching is receiving further study and the technique may find application for some of the more refractory raw materials.

Other techniques: Rotary drum mixers have been used in a number of operations. Most of the units were initially installed prior to the development of mechanical agitators suitable for relatively coarse slurries. Rotary drum units are also used in the strong acid cure leach at the Arlit mill in Niger. These units are used to blend the ground ore with concentrated sulphuric acid and a small amount of water.

Percolation leaching in concrete or wooden vats was used in a number of US uranium-vanadium operations. The calcine from the salt roasting step was first leached by percolation with water for vanadium recovery and then contacted with

¹ $6.9 \times 10^5 \text{ Pa} = 100 \text{ lbf/in}^2 (\text{g})$.

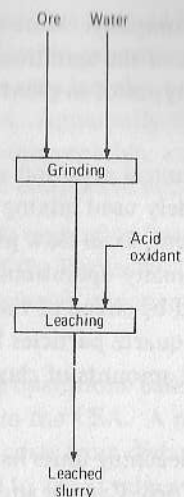


FIG.11. Series leaching circuit arrangement.

an acidic solution to extract the uranium. The Western Nuclear Corp. operated an acid cure and percolation leach at the Spook mill in Wyoming (Mashbir, 1974 [39]). The unconsolidated sandstone ore was crushed to about minus 10 mm, blended with concentrated sulphuric acid and a small amount of water to form pellets approximately 25 mm in diameter. These pellets were bedded into the percolation leach vats, and after curing for 24 hours, were leached with water by either downflow or upflow percolation.

Materials of construction: Acid-leaching operations have been conducted in rubber-lined steel tanks, wooden tanks, and fiberglass-reinforced plastic tanks. Acid brick linings have been widely used in many operations where abrasion was a problem. Most of the alkaline leaching operations have used mild steel construction, but stainless steel has been required to prevent corrosion in some autoclave operations.

Rubber-lined agitators have been used for most mechanical mixing units, but the design requirements have varied considerably. In-plant development studies after start-up have often been needed to determine the best type of rubber.

(b) Agitation leaching arrangements

A variety of circuit arrangements have been used for leaching uranium ores, but most are applications or combinations of the systems shown in Figs 11 and 12. Figure 11 illustrates the "series circuit" that is used in a high percentage of both the acidic and alkaline leach circuits. The ground ore slurry is fed to the circuit, and the material passes through a series of tanks without any recirculation.

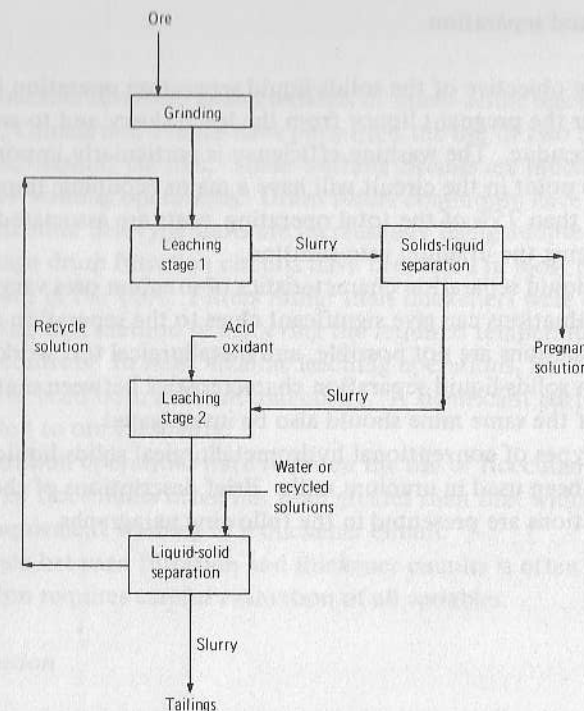


FIG.12. Two-stage countercurrent circuit arrangement (acid-kill circuit).

Circuits have contained from three to as many as 12 tanks in the series. The leaching reagents may all be added to the initial tanks or may be proportioned throughout the series. Figure 12 illustrates a "two-stage countercurrent circuit". This arrangement provides for leaching of difficult ores at relatively high acid concentrations in Stage 2 and for neutralization of the excess acid in the circulating solution by mixing with fresh ore in Stage 1. The circuit is particularly desirable when the uranium mineral requires a high acid concentration in the leach but the pregnant solution that would be obtained from a series circuit is too acidic for the subsequent concentration-purification step, particularly where IX is used.

Under these conditions the "two-stage circuit" can reduce both the acid requirement for the leach and the quantity of lime or ammonia needed for post-leach neutralization of the pregnant liquor. The two-stage circuit works best if dry grinding is used, but wet grinding can also be used if a dewatering stage is added between the grinding and leaching steps.

4.1.3. Solids-liquid separation

The primary objective of the solids-liquid separation operation in uranium mills is to recover the pregnant liquor from the leach slurry and to wash the soluble values from the residue. The washing efficiency is particularly important because any losses at this point in the circuit will have a major economic impact. In most operations more than 75% of the total operating costs are associated with mining the ore and bringing the uranium into solution.

The solids-liquid separation characteristics of uranium ores vary considerably. Mineralogical evaluations can give significant clues to the separation characteristics, but specific predictions are not possible, and metallurgical test work is mandatory. The variability in solids-liquid separation characteristics between materials from different parts of the same mine should also be investigated.

Nearly all types of conventional hydrometallurgical solids-liquid separation equipment have been used in uranium mills. Brief descriptions of the most common applications are presented in the following paragraphs.

(a) Thickeners

Countercurrent decantation (CCD) washing circuits using thickeners have been favoured in the USA and Australia. The development of effective flocculants such as the guar gums and the polyacrylamide reagents during the late 1950s was a key to the successful application of these circuits.

The washing efficiency of a CCD thickener circuit is a function of the number of stages, the amount of wash water introduced, and the per cent solids in the thickener underflows (the wash to underflow liquor ratio). Nearly all CCD circuits have been designed to achieve washing efficiencies of 98% or greater, but a variety of operational techniques and equipment arrangements has been used. For example, some operations have used four-stage CCD circuits with wash-water flows of about three litres of water per kilogram of ore passing through the circuit. Other mills have been designed with seven or eight washing stages; these circuits achieved the 98% washing efficiencies with only about one litre of water per kilogram of ore washed. Operational problems may be greater with the seven or eight stage circuits, but much higher grade pregnant liquors are produced and the reduction of liquor volume economizes water usage and reduces the size of the subsequent extraction equipment.

The underflow densities in any CCD circuit must be carefully controlled to achieve consistent washing efficiencies. Radiometric density controls and variable speed pumps have been used successfully in many mills.

(b) Filters

The favourable filtration characteristics of many acidic leach pulps in South Africa, Canada and France have permitted the use of two or three-stage drum filtration washing circuits. These washing circuits are much more compact than thickener washing operations. Drum filters commonly have been used, but currently horizontal belt-type units are increasingly being adopted.

Three-stage drum filtration circuits have been used in most of the alkaline leach operations in the USA. Filters rather than thickeners were chosen because they offered the best method of achieving the required temperature and solution composition controls. In most alkaline leaching operations, however, the filtration operations have been both difficult and costly. A significant part of the difficulties has been related to ore variability.

Most filtration operations have required the use of flocculants, and the reagent consumption of flocculants often has been greater than that which would be required for equivalent washing in a thickener circuit.

The choice between filtration and thickener circuits is often not clear-cut and the decision requires careful evaluation of all variables.

(c) Classification

Various types of classifier circuit have been used as washing systems in uranium operations. Most of these circuits have been used in conjunction with resin-in-pulp (RIP) ion-exchange operations. The classifiers produce a slime slurry from the leach pulp stream and wash the soluble uranium values from the sand fraction. Cyclone units, spiral classifiers, and combinations of the two have been used. Normally six or more stages of classification and washing have been required. The slimes stream contains 98% or more of the soluble uranium values, and this stream becomes the feed for the RIP system. In most instances the classification separation is made at about 40 μm (325 mesh).

Classifiers have also been used in conjunction with CCD thickener circuits; the classifiers wash the sand fraction and the slimes fraction is washed in the thickeners.

Clarification or polishing of the pregnant liquors is necessary before the solutions go to solvent extraction or fixed-bed (downflow) ion-exchange columns. Very small amounts of solids can cause serious emulsion problems during solvent extraction separations and can also collect on top of the resin bed and reduce the flow-rate through the fixed-bed ion-exchange columns. A wide variety of filters have been used for the polishing step; these include gravity sand filters, pressure sand filters, various types of leaf filter, and precoated-continuous-vacuum drum filters. Even with careful laboratory test work, it often has been difficult to develop good specifications for the clarification equipment. Most mills have used

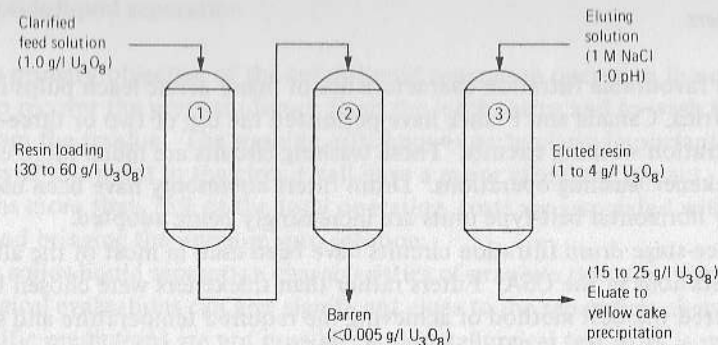


FIG.13. Three-column fixed-bed ion-exchange system.

various types of sand filter. Precoat drum filters are generally expensive but offer more flexibility. Mill managers and operators often comment on the difficulties and costs associated with these operations.

4.1.4. Concentration/purification operations

The development of resin ion-exchange (IX) and solvent-extraction (SX) techniques for separating uranium from leach liquors has been one of the major contributions of the uranium industry to hydrometallurgical technology. The uranium industry was the first to make extensive use of these procedures. Until the installation of the first uranium IX plants in the early 1950s and the construction of the initial SX circuits in about 1957, uranium separations had been based either on roasting and fusion reactions or on chemical precipitations. The introduction of IX technology both reduced operating costs and resulted in better product purity.

As discussed in Section 3, the recovery of uranium from acidic solutions by either IX or SX is based on the selective absorption of the $\text{UO}_2(\text{SO}_4)_3^{4-}$ anion. This uranyl sulphate ion is subsequently eluted or stripped from the loaded organic reagent by contacting with suitable concentrations of Cl^- or NO_3^- or other anions. Several systems have been developed to bring the pregnant leach solutions into contact with the exchange media. The following paragraphs discuss the development and application of these systems.

(a) Ion-exchange systems

The first IX systems used in the uranium industry were fixed-bed column units that were based on technology originally developed for water softening.

In these systems the feed solution flows downwards through a packed or "fixed-bed" of ion-exchange resin. The first units were installed in the early 1950s, and a number of these systems are still in operation. Several modifications of the "fixed-bed" system and a variety of operating procedures have been developed.

A number of continuous or semi-continuous countercurrent ion-exchange (CIX) contactors have also been developed, and are being increasingly adopted commercially.

Fixed-bed ion exchange: A simplified diagram for a conventional 3-column fixed-bed ion-exchange system is shown in Fig.13. Approximate uranium concentrations that are typical for solutions and resin in many operations are also indicated. The column operation is based upon chromatographic principles. When the pregnant liquor passes through the column, the resin at the top absorbs uranium and becomes "saturated" or fully loaded first. The liquor is gradually depleted in uranium so that the solution leaving the column is "barren" and contains essentially no uranium. As the solution flow to the column is continued, a point is reached where uranium begins to appear in the stream leaving column 1. This condition is termed "breakthrough" and normally is considered to be the point at which the uranium concentration in this stream is about 2% of the concentration in the feed liquor.

The flow through column 1 is continued until the resin in the column is fully loaded. The system must be designed so that this condition is achieved before "breakthrough" occurs from column 2. Column 1 is then taken off line for elution, and the freshly eluted column 3 is brought in as the trailing column. Four column systems have also been used in many mills. The fourth column is needed for some feed liquors because breakthrough from column 2 occurs before column 1 is fully loaded.

The resin in the conventional fixed-column system does not move and remains in the same column unit. A modification of the conventional column system has been developed in which the loaded resin is fluidized and transferred to a separate elution system. This technique permits more efficient utilization of column space and also makes the resin accessible for regeneration or other special treatments. This type of system sometimes has been designated as a "resin-transfer" unit.

Fixed-bed systems have been operated successfully in many mills, and the design criteria for the system is well established. The capital costs, however, are appreciable, because complex automatic valve systems are required and the columns are relatively large pressure vessels.

A significant investment in resin inventory is also needed. To maintain adequate solution flow through the columns at reasonable operating pressures, nearly completely clarified feed solutions are necessary. These requirements and other operating problems have led to the development of CIX systems capable of handling unclarified solutions and dilute slurries.

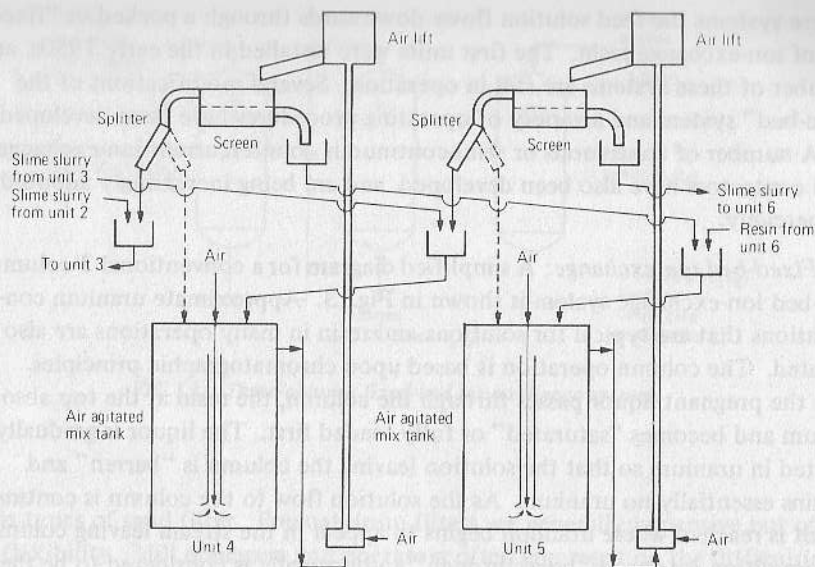


FIG.14. Screen-mix resin-in-pulp system (two units shown).

Continuous ion exchange (CIX): An approach to true countercurrent flow of resin and solution is achieved by several recent designs of equipment in which pregnant solution flows upwards through a vertical column divided into a number of compartments each holding a resin charge. The resin is fluidized by the upward flow of solution to achieve good contact for absorption of the uranium from solution and loading of the resin. Periodically the flow is interrupted or diverted to move the resin charge in each compartment downwards to the one below it. The net effect is that pregnant solution enters the bottom of the column and leaves the top as barren solution while freshly eluted resin enters at the top of the column and becomes loaded with uranium before discharge from the bottom. Elution of the loaded resin takes place in separate columns. CIX columns generally require less resin and can be operated with unclarified feed solutions containing entrained solids. Further description of the equipment is given in Section 4.3.5 which deals with the latest development in this field.

Resin-in-pulp (RIP) systems: Resin-in-pulp systems are designed to treat slime pulps, i.e. slurry mixtures of fine solids and pregnant solutions. The original designs, used in the USA, consisted of a number of baskets containing resin charges, suspended and moved cyclically up and down in long rectangular tanks or

banks through which the slime pulps are passed in series. These banks were operated in much the same manner as the fixed-bed units in that, as the resin in each lead bank becomes loaded with uranium, this bank is cut off from the slurry flow and eluted while freshly eluted resin banks are added on at the trailing end of the circuit.

A second type of RIP system, known as the screen-mix technique, achieves continuous flows of resin and pulp (see Fig.14). In this design, resin and slurry move in opposite directions through a series of agitated tanks (or stages). The resin or slurry discharged from each tank is mechanically separated, the slurry moving to a trailing tank and the resin to a preceding tank. The result is a continuous countercurrent flow of pulp and resin through the tank series to achieve resin loading and removal of uranium from the solution. A second series of tanks accomplishes elution with a similar countercurrent flow.

The resin and slime slurry contact in the air agitated mixing tank which has a retention time of about 20 min. The mixture of resin and slime slurry overflows and is air-lifted to a 65-mesh screen that separates the resin from the slime slurry. The separated slime slurry flows to the next unit on the right. The resin stream is split; part goes to the next unit on the left and the remainder recirculates to the mixing unit. This resin recirculation permits adjustment of the resin-slurry ratio in the mixing tank to give optimum resin retention times. All splitters operate on the same time cycle to ensure uniform movement of resin through the circuit this configuration has been operating for over 15 years.

Similar equipment arrangements have also been used for the elution circuits, but the number of stages is normally 10 to 12. The larger number of stages permits the production of relatively high-grade eluates containing 10 to 12 g U_3O_8 /litre.

A disadvantage, common to all ion-exchange systems that require separate devices linked to each stage for the transport of liquid and resin, is the possibility of inventory imbalances developed within the circuit. For example, the resin concentration in the pulp in a mixing tank can be maintained at a steady value only by the exact matching of the flow-rate of resin progressed to the stage plus the flow-rate of resin recycled to that stage and the flow-rate of resin leaving the stage, together with a similar balance on the slime flow-rates. The difficulty of measuring resin concentrations in a pulp directly precludes the use of control in an automatic feedback loop. Manual control demands a measure of both experience and vigilance on the part of the operators.

(b) Solvent extraction systems

Approximately 50% of the operating uranium plants today use SX as the primary means for recovering the uranium from solution. All these plants are acid leach operations, and nearly all use tertiary amines as the extraction reagent.

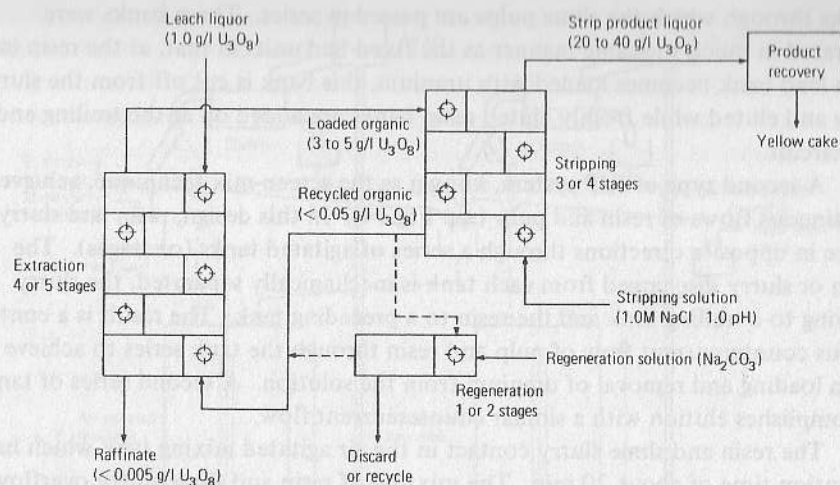


FIG. 15. Amine SX: Generalized flowsheet.

Multiple-stage mixer-settler systems are used as contacting and separation equipment throughout the industry. As discussed in Section 3, the reaction chemistry for amine solvent extraction is similar to that of resin ion exchange. The amine reagents are anion exchangers, and the principle uranium exchange ion is the $\text{UO}_2(\text{SO}_4)_3^{-4}$ anion complex. A generalized flowsheet for an amine extraction system is shown in Fig. 15. This flowsheet illustrates the sequence of operations together with the approximate uranium concentrations in the major process streams. Additional comments on the individual steps of the process will be presented in the following paragraphs:

Extraction: The first SX systems used separate mixers and settlers with considerable interstage pumping. Recent designs use integrated units in which multiple-stage mixers and settlers are contained within a single, compartmented rectangular vessel. The agitators are designed to provide both mixing and interstage pumping. The organic extractant is normally a mixture containing 4 to 7% amine, and 2 to 4% modifier (isodecanol or tridecanol); the balance of the mixture is kerosene or some similar petroleum product. The purpose of the modifier is to improve phase disengagement. The extraction kinetics are fast, and the required residence time in the mixers is normally less than 30 s. The mixing horsepower requirements per stage forming are usually in the range between 0.01 to 0.02 hp/gal (0.02 to 0.04 W/litre) of capacity. To promote better phase disengagement, most systems are operated with the organic phase continuous, i.e. aqueous droplets are dispersed in a continuous organic phase. This condition is achieved and maintained by recirculating part of the organic from the settler back to the

mixer. To achieve countercurrent flow, the organic and aqueous phases must be separated after each mixing step. This separation is primarily a function of the cross-sectional area of the settler. Settler capacities can vary considerably from operation to operation, but for organic-continuous operation the settler capacity is often in the range between 0.5 to 1.0 gal/min per square foot (0.18 to 0.36 litres/min per m^2) of settler area.

The number of countercurrent mixer-settler stages required for a given operation can be determined by a combination of batch extraction experiments and continuous countercurrent bench-scale experiments. Projections based on bench-scale countercurrent experiments have proven reliable. Agitation requirements have also been successfully projected from bench-scale experiments (Bellingham 1960 [40]). Prediction of settler performance can be more difficult because stabilized interface emulsions may or may not form in the bench-scale units. If these stabilized emulsions, which are often designated as interface crud, continue to build up at the interface, it may be necessary to shut down and clean out the material before the entire system becomes clogged. Variables such as small changes in the feed liquor clarification, fungus formation, and the slow polymerization of silicates can affect the interface crud formation. Ensuring complete clarification or polishing of the feed liquors is probably one of the best methods of minimizing crud formation.

Stripping: The equipment used for the stripping operations is usually identical to that used for the extraction step. In some instances the sizes of the mixers and settlers in the stripping section may differ from the corresponding equipment in the extraction section. For example, the required residence time for the stripping operation may be nearly double that of the extraction step; and the mixing horsepower requirement may be tripled. It is often possible, however, to use the same size equipment for both the extraction and stripping systems because the flows in the stripping system are smaller.

Chloride solutions have been used in many stripping operations, but other agents such as nitrates, sulphates, carbonates, hydroxides, and dilute acids can also be used. A controlled pH technique employing a 1.5M $(\text{NH}_4)_2\text{SO}_4$ solution has been widely used in both South Africa and the USA. In this system the pH is normally held in the range between about 3.5 to 4.2 by the addition of ammonia. Below this pH range the stripping efficiency decreases, and at higher pH levels hydrolysis of the uranium is likely. When hydrolysis occurs troublesome emulsions can form.

Regeneration: The solvent is generally purified before being returned to the extraction section. This is particularly important when the leach solution contains molybdenum, which is strongly extracted and is not removed effectively by conventional chloride strip solutions. The molybdenum concentration in the organic phase will continue to increase until gummy precipitates start to form at the

organic-aqueous interface. These precipitates can seriously interfere with the operation of the system. The molybdenum concentration is controlled by bleeding part of the stripped organic through a regeneration system, where it is contacted with a heated 10% Na_2CO_3 solution. This treatment removes the molybdenum; recovery of the molybdenum from the carbonate solution is also possible. The amine is converted to free base in this treatment as that of the anions are also removed.

(c) Combination of IX and SX

In many plants the combination of IX and SX is used. This plant is termed the Eluex process in some countries and the Bufflex process in others. In this system the uranium is absorbed from acidic leach solutions by ion-exchange resins, the resin is then eluted with a 10% H_2SO_4 solution with tertiary amine solvent. The controlled pH stripping technique is used to remove uranium from the loaded organic and produce a pregnant solution for the precipitation of yellowcake. The Eluex/Bufflex process can both decrease reagent costs and eliminate the introduction of extraneous ions such as chlorides or nitrates. Elimination of the extraneous ions permits more recirculation of tailings solutions to the leaching circuits, thereby reducing environmental problems.

(d) Comparison of the IX and SX systems

Choosing between resin ion exchange and solvent extraction for a given uranium milling operation is often a complex decision. Many factors must be considered that range from ore mineralogy to environmental constraints. Specific experimental studies and verifications are almost always required, and the final decision may be based on rather subtle considerations. An important consideration is related to differences in the flexibility of the two systems. For example, the continuous ion-exchange systems can handle unclarified leach liquors and dilute slurries, whereas essentially complete clarification of the feed liquors is required for solvent extraction. This can be particularly important for ores that contain clay-like materials. IX may also be favoured over SX when large volumes of leach liquors with low uranium concentrations are to be treated, because solvent losses are primarily related to the volume of solution handled. The greater fire hazards associated with solvent extraction circuits should also be considered.

For some operations, the SX system may be preferred because it can treat acidic feed solutions with lower pH values than can be accommodated by IX and this can significantly increase the flexibility of the leaching circuit operations. Also SX is generally more selective than IX and, therefore, high yellowcake purities are often easier to achieve.

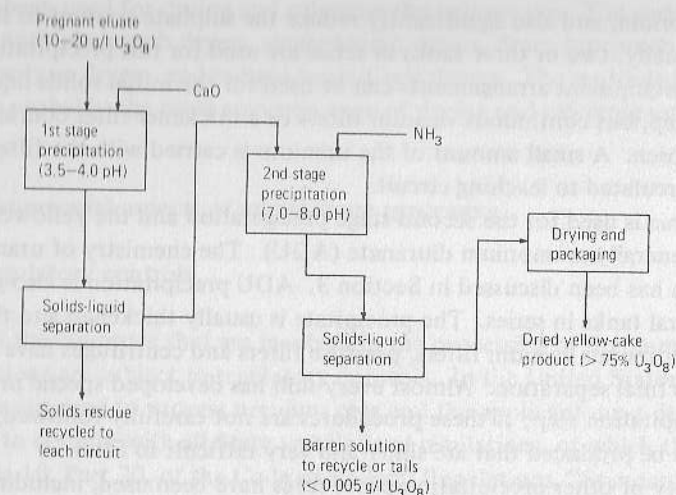


FIG.16. Continuous two-stage yellowcake precipitation circuit.

A variety of experimental studies and economic evaluations are required for optimum decisions, and extrapolations from other operations may be misleading. Both IX and SX should be considered for new operations or expansions.

Two recent studies have appeared on this topic. Wentzler and Neil (1978 [41]) compared CIX-Eluex with SX and concluded that there is economic incentive for considering the CIX-Eluex for any new operation. Brown and Hayden (1978 [42]) concluded that, for concentrations greater than 0.9 g U_3O_8 /litre, SX is favoured and below 0.35 g/litre, IX is favoured.

4.1.5. Precipitation and final product preparation

Even the yellowcake precipitation step can be affected indirectly by the ore mineralogy because impurities dissolved from the ore that have passed through the subsequent steps may require separation during the final processing operations. These impurities may be elements such as molybdenum, vanadium, and iron, contained in the ore, or components such as Na^+ ions introduced with the eluting or stripping solutions. Over the years, specifications for yellowcake concentrates have gradually become more restrictive; and many mills have found it necessary to modify their product precipitation procedures. In most instances single-stage or batch operations have been replaced with multiple-stage precipitation circuits. Figure 16 shows a general schematic flowsheet for a continuous two-stage precipitation circuit that uses lime for the first-stage precipitation and ammonia for the final operation. The first-stage precipitation can remove iron, aluminium,

titanium, thorium, and also significantly reduce the sulphate content to the liquor. Normally, two or three tanks in series are used for this precipitation. A variety of equipment arrangements can be used for the initial solids liquid separation step, but continuous vacuum filters or a thickener-filter combination often are chosen. A small amount of the uranium is carried with the filter cake which is recirculated to leaching circuit.

Ammonia is used for the second-stage precipitation and the yellowcake product is generally ammonium diuranate (ADU). The chemistry of uranium precipitation has been discussed in Section 3. ADU precipitation is also generally done in several tanks in series. The precipitate is usually thickened and then filtered. Continuous vacuum filters, pressure filters and centrifuges have been used for this final separation. Almost every mill has developed special procedures for the precipitation step; if these procedures are not carefully followed, precipitates can be produced that are slimy and very difficult to filter.

A variety of other precipitation procedures have been used, including:

- Single-stage precipitation with ammonia
- Single-stage precipitation with MgO
- Single-stage precipitation with NaOH
- Two-stage ammonia precipitation with redissolution between stages
- Two-stage precipitation with MgO and ammonia
- Precipitation with H_2O_2 .

The precipitation with H_2O_2 is carried out by adjusting the solution pH to about 3.5 and then adding the hydrogen peroxide. This procedure is generally more costly than other techniques, but products with grades of 98% U_3O_8 or greater have been produced. The procedure is a good method for producing yellowcake with very low molybdenum and vanadium contents.

Alkaline leaching circuits and some carbonate stripping operations produce liquors in which the uranium is present as stable anionic carbonate complexes. The uranium is precipitated from these alkaline leach liquors as sodium diuranate or polyuranates by adding sodium hydroxide (caustic) to the pregnant leach liquor. The sodium content of the product is high but can be reduced by washing the cake with an ammonium sulphate solution or by redissolving the cake and making a second precipitation with ammonia. The carbonate strip liquors usually are precipitated by acidifying to destroy the carbonate and then adding ammonia to precipitate an ammonium diuranate yellowcake.

The yellowcake products are usually dried at 400 to 500°C. The drying temperatures must be controlled because temperatures above 600°C can produce refractory compounds that will not dissolve in subsequent refining and conversion steps necessary for the production of UF_4 . Yellowcake products dried at low temperatures tend to be relatively light and fluffy. Considerably more packing and shipping space is required for this type of material. A wide variety of equip-

ment has been used for drying and calcining the yellowcake. The various types include: multiple-hearth dryers, single-hearth dryers, drum-type units, Holo-Flite screw-type dryers, and radiant-heated belt dryers. The multiple-hearth furnace is probably the most common type of drying and calcining unit used in the industry.

4.2. Environmental aspects of uranium ore processing

4.2.1. Regulatory controls

In all the countries that are involved in the processing of uranium ores, the operations are subject to regulatory controls. In the United States of America, a license is required to process uranium ores and the applicant must demonstrate an ability to comply with all State and Federal regulations, of which the principal one is Title 10, Part 20, of the Code of Federal Regulations, "Standards for Protection Against Radiation". The regulations specify radiation exposure limits in the milling facility and the maximum permissible levels of radioactivity in the process wastes and dusts.

Radiation protection standards in Canada are the responsibility of the Atomic Energy Control Board (AECB).

4.2.2. Radioactivity

The basic unit of measurement of radiation activity is the Becquerel (Bq), which is defined as equal to 1 dis/s. Each tonne of ore entering a uranium mill has a radioactivity of about 1.85×10^5 Bq. This is comprised of ^{238}U plus 14 radioactive decay isotopes, of which four (^{234}U , ^{230}Th , ^{226}Ra and ^{210}Pb) are sufficiently long-lived to contribute to the radioactivity of tailings.

Three measurements for adsorption of ionizing radiation are the roentgen, the rad, and Joules/gram (J/g). The roentgen is a unit of gamma-ray exposure, whereas a rad (radiation absorption dose) is the radiation energy absorbed, for example, by the human body. Radiation exposure for control purposes is measured in J/g (roentgen equivalent man), which, for practical applications, is usually equal to the dose of radiation received by the human body. Conversion from rads, or roentgens, to J/g depends upon the type of radiation (alpha, gamma).

4.2.3. Airborne wastes

Adequate ventilation and dust suppression are required for control of airborne radioactivity. The mill environment is monitored for concentrations of radioactivity and personnel-type monitoring devices, such as film badges, are required where workers may be exposed to continuous radioactive dosage.

Conventional dust collection equipment is usually suitable for reducing radioactive dusts. However, special attention is needed in the yellowcake drying, handling and packaging areas. Various gases and vapours evolved during treatment of some ores may be hazardous and require the installation of additional ventilation. Acid vapours, ammonia, sulphur dioxide and other gases may be injurious to health and require attention.

4.2.4. Liquid wastes

An objective for new milling facilities is total containment of all the liquid wastes. While this is only possible where high evaporation rates prevail, the practice of recycling solution from the tailings-area decant pond to the process reduces the amount of new water required; the amount which can be recycled is limited by its effect on the process metallurgy.

One to five tonnes of waste solution are generated for each tonne of ore processed. The amount which cannot be returned to the process is evaporated, or where this is not entirely possible, the excess liquids are released to surface watercourses after suitable treatment to reduce radioactive nuclides and other contaminating elements to the desired levels.

A variety of chemicals eventually go to the tailings disposal area; these may include sulphuric acid, sulphates, carbonates, chlorides, nitrates, ammonia, lime, cyanides, polyacrylamide flocculants, fuel oils, and other organics such as amine solvents, kerosene, and alcohols.

In alkaline leaching, the barren liquid is recovered and reused. The leaching residues are washed and the final wash solution, transported to the tailings pond with the solids, may be contaminated by radioactive nuclides and chemicals. Experience has shown about 1–3% of the radium but virtually no thorium dissolves in alkaline leach circuits. Much of the dissolved radium, however, is precipitated with the yellowcake.

In acid leaching, less than 1% of the radium but about 50% of the thorium present in the ore feed become solubilized and report in the solutions sent to waste disposal. The main process for treatment of acidic mill effluents is neutralization with limestone and/or lime to: neutralize H_2SO_4 ; precipitate sulphate and heavy metals; and remove some radionuclides.

Further decontamination of the neutralized effluents, usually decant liquor from the tailings pond, is by barium chloride treatment. Radium-226 can be co-precipitated with barium sulphate by addition of $BaCl_2$ to clarified liquor in the presence of excess sulphates at pH 8–9. Sufficient settling-pond area must be available for accumulation of the insoluble precipitate; up to 60 days is provided for this purpose. The addition of flocculants is beneficial in obtaining the desired levels of total radium.

Alkaline effluents may also be partially decontaminated of their radium content by barium chloride treatment.

In Canada, an Ontario guideline limits the ammonia content to 1 ppm in any effluent discharged to a watercourse. Agnew Lake Mines Ltd. use ammonia in the solvent extraction and precipitation circuits and, in order to meet the requirement, have incorporated a lime-boil stage for treatment of the bleed stream. The ammonium sulphate solution formed is reacted, at temperature, with milk-of-lime to release ammonia and precipitate gypsum. The ammonia is steam-stripped from the resulting slurry in a counter-current series of agitated vessels and condensed as aqua ammonia. The capital and operating costs of such a system do not justify its use on the basis of uranium recovery alone.

At some locations seepage of solutions from tailings ponds into the ground accounts for 80% or more of the water loss. For new disposal areas, the regulations require that the area overlying permeable strata must be sealed to prevent rapid seepage. Although contaminants are usually removed from percolating solutions by natural ion exchange, adsorption, and chemical precipitation processes which occur in the earth, regulations require that monitor wells be located in the vicinity of the tailings disposal areas.

Evaporation is practised in the western USA to control liquid waste disposal. From 0.6 to 1.7 ha of pond surface are required to evaporate one tonne of water per hour, depending upon the local conditions.

4.2.5. Solid wastes

The leach residues must be deposited in a suitable waste retention system which does not present a hazard. In certain instances the discharge of tailings has been made into isolated lakes or to underground mines as backfill. The most significant problem in uranium-mill solid-waste handling, which differs from other ore-processing operations, is the need to control the release of radioactive materials from the tailing impoundment area. A high degree of safety is required against accidental release of solids and control of seepage into the underlying strata.

The mill tailings contain about 70% of the total radioactive constituents originally present in the ore. With time, radon and its daughters begin to form in the tailings from its ^{226}Ra parent.

For an operational mill, tailings deposited in a properly located, engineered and managed waste retention system do not present a significant hazard. However, after the mill has been decommissioned, some problems may result:

The slimes portion, containing the majority of the radioactivity, may remain fluid and could, by hydrostatic pressure on an embankment, cause a failure, especially if weakened by erosion and seepage;

Loss of wind-blown fines;

Percolation of waters causing contamination of the sub-strata by radio-nuclides; and

Continued generation of acid from any sulphides.

Revegetation of a tailings pond has been found to be a suitable means to stabilize the surface, but often this has been difficult and costly to initiate and sustain. Other methods of surface stabilization include spraying with petroleum products to form wind-resistant crusts.

4.3. Energy requirements and processing costs

Because of wide variations in the mineral composition of uranium ores generalized predictions are difficult and each situation must be evaluated separately. The information presented in the following paragraphs does not apply to any specific operation but is intended to give a general feel for the energy and cost distributions that may be encountered in uranium ore processing.

4.3.1. Energy requirements

Energy requirements for the processing of uranium ores are becoming increasingly important. Not only are the thermal and electrical energy requirements significant, but the cost of reagents and supplies is directly affected by the energy required to produce these materials. Comparison of the total net electrical, thermal, and chemical energy requirements can be useful in comparing alternative techniques.

Estimates of energy requirements for several different uranium ore processing circuits were made during a study sponsored by the US Department of Interior, Bureau of Mines, at the Battelle Columbus Laboratories (1975 [43]). Data from the Energy Use Patterns report were used here to develop a comparison of the energy requirements for the acid leach and alkaline leach flowsheets shown previously in Figs 9 and 10. Calculations for the comparison were based on a feed ore grade of approximately 0.2% U_3O_8 , and the overall recovery was assumed to be 95%. The H_2SO_4 consumption for the acid leaching operation was assumed to be 25 kg/t ore processed.

Most of the energy consumption data in the Energy Use Patterns report was used directly except for the yellowcake drying requirements, which appeared to be high. The thermal requirements for this operation were reduced to conform with data obtained from the US Department of Energy.

The Energy Use Patterns report expresses energy requirements in terms of Btu/ton U_3O_8 in the product. Both electrical and direct fuel requirements are expressed in this form, and Btu values were also assigned to major reagents and supplies such as grinding media. Table IV lists these energy values. The Btu value

TABLE IV. NET ENERGY CONTENT OF THERMAL, ELECTRICAL, AND CHEMICAL ENERGY SOURCES

Energy source	Unit	Joules $\times 10^6$
Electricity	(kW·h) J	(3.6) 1.0
Natural gas	(ft ³) m ³	(1.055) 37.25
H ₂ SO ₄	t	965
NaOH	t	34 887
NH ₃	kg	9.3
CaO	t	9884
NaClO ₃	t	67 158
Oil and grease	litre	41.8
Kerosene	litre	37.6
Grinding media	kg	8.4

indicated for each chemical or supply is the amount of energy required to produce that product from natural raw materials in the ground. These values were used to determine the overall energy consumption.

The calculated total net energy input required to produce 2000 lb U_3O_8 by either acid or alkaline processing is shown in Table V. The comparative energy distribution for the various unit operations is also indicated.

The energy requirement per unit of uranium product for the alkaline circuit is nearly double that required for the acidic circuit. The relative energy required by the various operations, however, is similar. In both processes, nearly 80% or more of the energy used is associated with operations directly related to the quantity of ore being processed (see Fig.4). The largest single energy input is required for the leaching step. Even though the reagent addition to the acid leach circuit was set at only 25 kg H_2SO_4 /t ore and heating of the acid leach pulp was assumed, over 55% of the leaching energy requirements was associated with the acid addition. The total energy input is, therefore, quite sensitive to the acid consumption requirements of ore. Nearly 75% of the energy input to the alkaline leaching step is associated with the processing heat requirement. In both circuits, however, the actual energy content of the leaching chemicals is similar.

TABLE V. ENERGY REQUIREMENTS AND DISTRIBUTION FOR URANIUM ORE PROCESS

Item	Type of circuit	
	Acid leaching	Alkaline leaching
Net energy requirement (J/1000 kg $U_3O_8 \times 10^3$)	350 to 450	700 to 800
Per cent energy distribution (by unit operations)		
Crushing and grinding	26	31
Leaching	38	42
Solid-liquid separation	10	5
Uranium extraction (SX or IX)	11	
Stripping, precipitation, and drying	15	22
	100	100

TABLE VI. DISTRIBUTION OF THERMAL, ELECTRIC AND CHEMICAL ENERGY REQUIREMENTS

Type of energy	Per cent distribution of energy:	
	Acid leaching (SX or IX)	Alkaline leaching
Thermal energy (process heat)	32	43
Electrical energy	30	29
Chemical energy (reagents, etc.)	38	28
	100	100

In general, energy relationships outlined above do apply to a significant number of US sandstone and limestone ores, but reagent requirements can vary considerably for other ores. A determination of the leaching reagent requirements for each new ore source must be made before realistic total energy needs can be estimated.

Table VI shows a distribution of the overall thermal, electrical, and chemical requirements for the acid and alkaline processes.

In general the thermal energy requirements are greater for alkaline leaching but the chemical energy needs for acid processing are the highest. The difference would be even larger for operations where acid additions of greater than 25 kg/t ore are used.

In some instances energy requirement estimations can be a more useful tool for process comparisons than cost estimates because costs can vary considerably in different parts of the world. An energy balance can also be used to predict the effect of future energy cost changes.

4.3.2. Capital and operating costs

Capital and operating cost data are specific to the location of the uranium ore processing plant because of major cost variations in different parts of the world. Capital costs related to the individual unit operations can generally be used more widely than operating costs because methods for allocating the indirect and fixed portions of operational costs to the individual unit operations can vary greatly in different organizations. Even when actual data from different operations are available, operating cost comparisons can be confusing unless the procedures used for allocating the indirect costs are known. Merritt (1971 [44]) lists the following items as direct and indirect operating costs, but these generalized divisions vary from company to company:

Direct operating costs:

- Operating labour
- Maintenance labour
- Operating supplies (grinding media, chemicals, lubricants, etc.)
- Maintenance supplies
- Utilities (power, fuel, water).

Indirect operating costs

- Overhead labour (management, supervision, clerical, laboratories, accounting, warehousing, watchmen, etc.)
- Supplies (health, safety, clerical, water treatment for the general facility)
- Utilities (for the general facility)
- Payroll overhead (unemployment insurance, social security, paid vacations and sick leave, public liability insurance, unscheduled overheads, etc.).

Fixed costs

- Amortization cost of the investment
- Taxes and insurance.

Even though generalized capital and operating costs can seldom if ever be used directly, this type of information can serve as a useful guide for development

TABLE VII. PERCENTAGE DISTRIBUTION OF EQUIPMENT COSTS BY UNIT OPERATIONS

Unit operation	Per cent of total equipment cost:				
	Acid circuits:				Alkaline circuit
	SX	IX column	RIP basket	Eluex	
Crushing and grinding	26-45	26-45	28-48	24-41	31-49
Leaching	5-15	5-15	5-16	5-14	10-17
Solid-liquid separation	24-38	22-36	10-16	8-12	28-36
Uranium extraction (SX, IX, RIP)	14-16	15-17	26-32	35-45	
Product precipitation and drying	5-9	6-10	6-10	5-9	7-16

programmes on new ore sources or for modifications to existing operations. Cost evaluations are an integral part of any prudent development programme and the cost studies should be continually upgraded as new information becomes available. An examination of generalized cost data can be particularly helpful to metallurgists, mining engineers, and geologists. The unit operations that tend to have high capital and operating costs should receive the most careful scrutiny. The objective of the following section is to present a brief summary of capital and operating cost data that can help to develop an appreciation of cost relationships between the various unit operations.

The capital cost of a uranium-milling operation is related primarily to the amount of ore that is being processed, but the mineralogical characteristics of the ore also can have an appreciable influence on costs of the individual unit operations. Merritt has presented information on the percentile range of process equipment costs by unit operations, and these data were used to develop the percentage distributions shown in Table VII.

The data in Table VII are for equipment costs only and do not include the cost of foundations, installation, piping, etc. The percentages, however, do reflect the amount of equipment that is required for the various unit operations. Examination of the data shows that about 90% of the equipment costs are related to the unit operations that are directly affected by the quantity of ore being fed to the mill. The distribution range for each unit operation is determined primarily by variations in the mineralogical characteristics of the ore being treated. For example, unconsolidated sandstone ores will need only minimal crushing and

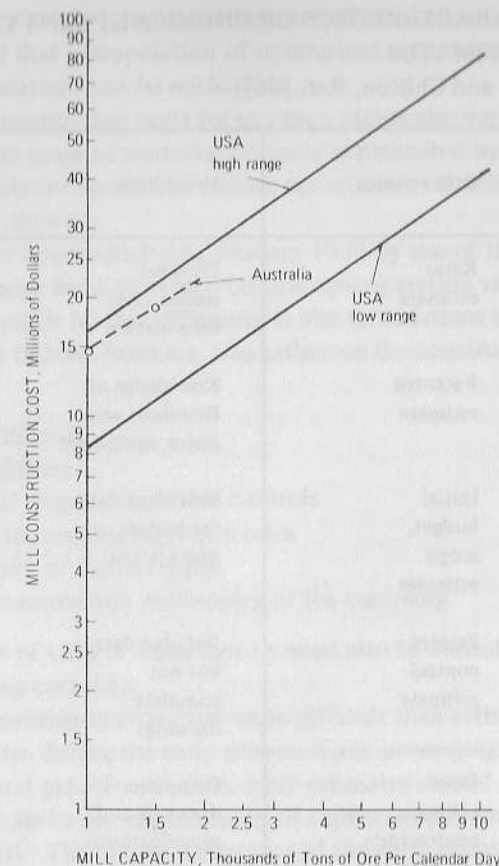


FIG.17. Uranium mill construction costs (adjusted to 1 January 1978).

grinding, while hard sandstones and limestone ores may require nearly twice as much equipment. Leaching equipment costs vary over an even wider range because optimum leaching times, temperatures, and degree of agitation show major variations. Significant differences in leaching requirements can occur between sandstone ores from different locations, and even greater variations can be expected between different types of ore. The amount and cost of equipment required for the solid-liquid separation is usually a function of the amount and type of fines in the ore. If significant percentages of montmorillonite-type clays are present, the solids-liquid separation operations may be difficult and complex. The range in equipment costs for the uranium extraction operation (SX and IX) is smaller, because these operations are less affected by differences in the mineralogical composition of the ore. The data in Tables V-VII illustrate again that each ore

TABLE VIII. CHARACTERISTICS OF CHEMICAL PLANT CAPITAL-
INVESTMENT ESTIMATES

(Taken from Perry and Chilton, Ref. [46])

Type of estimate	Other names	Usual basis	Minimum error (%)
Order of magnitude	Ratio estimate	Previous similar cost information	$> \pm 30$
Study	Factored estimate	Knowledge of flowsheet and major equipment	± 30
Preliminary	Initial budget, scope estimate	Sufficient data for budget preparation	± 20
Definitive	Project control estimate	Detailed data, but not complete drawings	± 10
Detailed	Firm estimate, contractor's estimate	Complete drawings and specifications	± 5

TABLE IX. RELATIVE COST DISTRIBUTIONS FOR ACID LEACH-SX
PROCESSING

Unit operation	Equipment cost (%)	Capital cost (%)	Operating cost (%)
Crushing and grinding	28	29	16
Leaching	9	9	46
Solid-liquid separation	46	44	22
Solvent extraction	11	12	11
Product precipitation and drying	6	6	5

source can present a nearly unique combination of mineralogical and processing characteristics, and that extrapolation of milling and processing equipment costs based on other operations can be misleading.

Overall mill construction costs for uranium plants also vary considerably. Figure 17 shows the range of costs for US mills as presented by Merritt. A plot of construction costs for Australian milling operations as developed by Hartley (1972 [45]) is also shown.

The costs have been adjusted to January 1978 by use of the CE Plant Cost Index. The cost range for a given size US uranium operation varies by nearly a factor of 2.0, and much of this difference is due to variations in the ore mineralogy. Other factors, however, also influence the construction costs; these include:

- Location of the mill
- Climatic conditions
- Environmental requirements and controls
- Creativity of the engineering contractor
- Risk philosophy of the company
- The general construction philosophy of the company.

The influence of each of these factors must also be considered when evaluating and comparing cost data.

Predicting operating costs is even more difficult than estimating capital costs. Developing estimates during the early phases of any processing study is important, but the accuracy and precision of these early estimates should not be overrated. Approximate error levels for various types of capital investment estimates are shown in Table VIII. The absolute accuracy of the early estimations is limited, but these estimates are useful guides for directing the experimental investigations.

Very little data have been published on actual operating costs for individual unit operations. Presenting this type of information is particularly difficult because of the details that must be shown to make the analysis meaningful. As mentioned previously, even the methods for allocating indirect costs vary considerably for different companies. It is possible, however, to develop a general feel for operating cost relationships from the limited data that have been published. Order-of-magnitude costs for the various unit operations of an acid leach-solvent extraction mill were derived from the data used to prepare cost estimates presented by Rosenbaum and George [47]. A summary of the equipment cost, total capital cost, and operating cost distributions for a 2000 t/d mill processing a 0.2% U_3O_8 ore is shown in Table IX. The estimated overall recovery was approximately 95%.

Nearly 70% of the total operating cost is associated with the leaching and solids-liquid separation operations. Both the steps are dependent upon the amount of ore being treated, but are also strongly influenced by the mineralogical composition of the ore. The cost distribution for these two steps also reflects the

importance of the capital investment for the solids-liquid separation and the reagent costs for the leaching operation.

Many factors influence the accuracy of capital and operating cost predictions, but defining the process metallurgy is fundamental. This definition can best be achieved by using a blend of metallurgical mineralogy, metallurgical experimentation, and past experience. When designing a mill, it is extremely important that close communication be maintained between the design engineers and those who have participated in the metallurgical development studies. Nearly every experienced metallurgist can cite several examples of problems that developed when this communication was not maintained throughout the design process.

5. NEW DEVELOPMENTS IN PROCESSING TECHNOLOGY

5.1. Introduction

The modern uranium ore processing flowsheet is made up of a combination of unit operations, all of which have been made more efficient over the years to achieve low operating costs, relatively high extraction, and satisfactory product purity. Each of these process steps should however be examined with regard to possible additional improvements, along with such processes as in-situ leaching and other novel techniques for the treatment of low-grade and refractory uranium ores.

5.2. Preconcentration and sorting

Physical beneficiation processes for upgrading or rejection of reactive gangue in the treatment of lower grade and refractory uranium ores and tailings continue to be a challenging area for study.

To achieve high recoveries and selectivity at the same time is particularly difficult with the mineral assemblies and textures of most uranium ores, a feature they share with fine low-grade tin ores. Work on the fundamental properties of fine particles to suggest possible processes may therefore prove rewarding.

Preconcentration has been used successfully in recent years in France, South Africa and Canada, and a considerable amount of work on various methods continues to be done in these countries. However, potential advantages of techniques such as photometric and radiometric sorting, heavy media separation, gravity concentration, wet high-intensity magnetic separation or flotation remain problematical.

Nevertheless, the development of a successful upgrading technique could greatly improve the capacity of a uranium mill without proportional increases in the mill cost.

5.3. Crushing and grinding

In the United States in particular, the introduction of semi-autogenous grinding for uranium ores has resulted in economic savings. The greatest advantage is in the elimination of the crushing operation and the problems encountered with handling and storing wet, sticky or frozen ore. A further significant advantage of semi-autogenous grinding is that dust (and radon gas) emissions are confined to one area and can be better controlled.

Chunks as big as 60 cm may be fed directly to the mills which operated with from 2–4% ball charge at a density of about 70% solids. Sandstone ores tend to literally fall apart very rapidly in the semi-autogeneous mills and the action seems to be one of scrubbing more than actual grinding. In fact, with a grate discharge, it is difficult to keep the mill loaded.

One problem with autogenous grinding so far has been in size classification of the product, since wood fibres in the pulp tend to clog stationary screens and cyclones. With screens, this can lead to prohibitive circulating loads. Base metal mills have used the DSM screen while uranium mills, planning to use autogenous grinding, are considering horizontal vibrating screens for sizing. Open-circuit operations also are feasible since not too many ore particles are normally found at or near the limiting size of the mill grates, which is usually 60 to 90 mm. However, these occasional coarse particles must be dealt with in the following leach tanks and thickeners and this can be a problem.

In one existing installation and in at least two planned milling circuits, a semi-autogenous mill is followed by a conventional rod or ball mill. This circuit retains the economic benefit of elimination of the crushing circuit, but maintains the operating reliability of size reduction for maximum uranium liberation and avoids tramp oversize particles from entering the leach circuit.

5.4. Leaching

5.4.1. Equipment and techniques for "in-plant" leaching

Leaching in most mills has become relatively straightforward, barring upsets due to frothing, mechanical problems, or an ore with unexpected characteristics. However, most studies on the leaching of uranium ores have been empirical, and it is possible that improvements may be made in leaching efficiency or in the treatment of difficult ores, if a more fundamental approach is adopted with special emphasis on the relationship between the mineralogy of the ore and its leaching response.

Ores with high carbon content often prove difficult to handle and sometimes unprofitable. High lime and high clay ores are troublesome and usually require special treatment. Roasting may be useful to improve ore characteristics but

little is known about the reactions involved or how to make them proceed to best advantage.

Future studies should examine the oxidation and dissolution reactions, and the effect of the many system variables on the kinetics of these reactions.

The extraction of uranium by acid pressure leaching was first investigated in South Africa in the 1950s when a small continuous pilot plant was operated by the Anglo-American Corporation. Although extractions of up to 94% uranium were achieved, this approach was put on ice because it was uneconomic at the time.

The recent increase in the prices of uranium and gold, and in the cost of sulphuric acid, has renewed interest in the process and a collaborative investigation by the uranium industry is being undertaken on pilot-plant scale at one of the gold mines. The aim is to test the reliability and performance of the equipment in full-scale operation, to establish a correlation between the results of bench-scale and pilot-scale tests, and to provide the mining companies concerned with practical information from which they can evaluate the use of pressure leaching in their own areas. Economic studies suggest that operating costs are comparable with those of conventional leaching, although capital costs are higher. It is expected that design work on one or more uranium treatment plants incorporating pressure leaching will be started soon.

The use of a pipe reactor, termed a tube digester, is being investigated on a pilot-plant scale in South Africa for the acid pressure leaching of uranium ores.

5.4.2. *In-situ leaching techniques*

5.4.2.1. Heap leaching

A significant number of moderate-scale heap-leaching operations and experiments have been performed, but relatively little information on the results has been published, particularly concerning uranium recovery efficiencies. In the United States most of the heap-leaching operations have been used to extract the uranium from material which is below cut-off, feed-grade ore at operating mines and mills. A number of companies and organizations throughout the world are re-evaluating heap leaching as an extraction method for relatively small deposits. The mineralogical composition of the ore has a major influence on the rate at which the leach solutions can be percolated through a heap and the method described by Kaiman [20] is useful but it has not yet been possible to predict the response of a given ore from mineralogical data alone. Experimental testing is mandatory.

The US Department of Interior, Bureau of Mines, has undertaken studies to gather fundamental and comparative data on the heap-leaching characteristics of ores from New Mexico, Wyoming and Colorado. The overall objective of the

work was to develop a technique for evaluating the heap-leaching potential of any given ore.

Heap leaching is potentially an attractive low-cost method since it eliminates much of the cost of crushing, grinding, leaching and liquid-solid separation, all of which are high capital and operating cost items. Like beneficiation the method does not yet achieve high recoveries. However, the problem would be less if techniques for forecasting recoveries with greater accuracy could be established. It is apparent that more work is needed to develop improved techniques for heap construction and operation, more efficient solution to ore contact, effective controls over pH and oxidation levels, and means to avoid interference from gypsum and clays. There must also be provisions for dealing with environmental problems, including what to do with the heaps after leaching is completed. Work over the past decade has resulted in a better understanding of the role of bacteria in heap leaching and new methods are being suggested using external fermentors to generate ferric iron and acid externally, thus alleviating the problem of oxygen supply. There are bound to be improvements in these techniques.

5.4.2.2. Solution mining

The interest in solution mining of uranium orebodies has increased significantly in recent years. The unconsolidated sandstone deposits of Wyoming and Texas have been the prime target for this development in the USA. Initial investigations were primarily acid leaching operations, but most of the current operations use various alkaline leaching techniques, ammonium carbonate solutions being particularly favoured.

Certain prerequisites are necessary for successful application of the technique. The first is that the mineralogy of the deposit must be amenable to the process; dissolution of minerals other than those containing uranium is usually important with regard to their possible influence on permeability of the strata. In Texas the uranium is present as uraninite and coffinite, and the deposits generally conform to a characteristic "roll-front" shape. The deposit must be underlain and overlain by non-permeable strata (e.g. shale) and must lie below the water table. In addition to those operations in Texas, pilot operations have been conducted in Wyoming. The Wyoming sandstones are reported to be generally less permeable than South Texas sandstones.

The lixiviant is delivered to the orebody through injection wells and the pregnant solution withdrawn through recovery wells. In practice, pumping pressures cannot exceed the collapse pressure of the ground structure. The relative physical disposition of injection and recovery wells may vary according to the local geological features.

A typical well pattern and vertical section of a deposit, one of the first in operation, is shown in Fig.18. In this arrangement monitor wells were established,

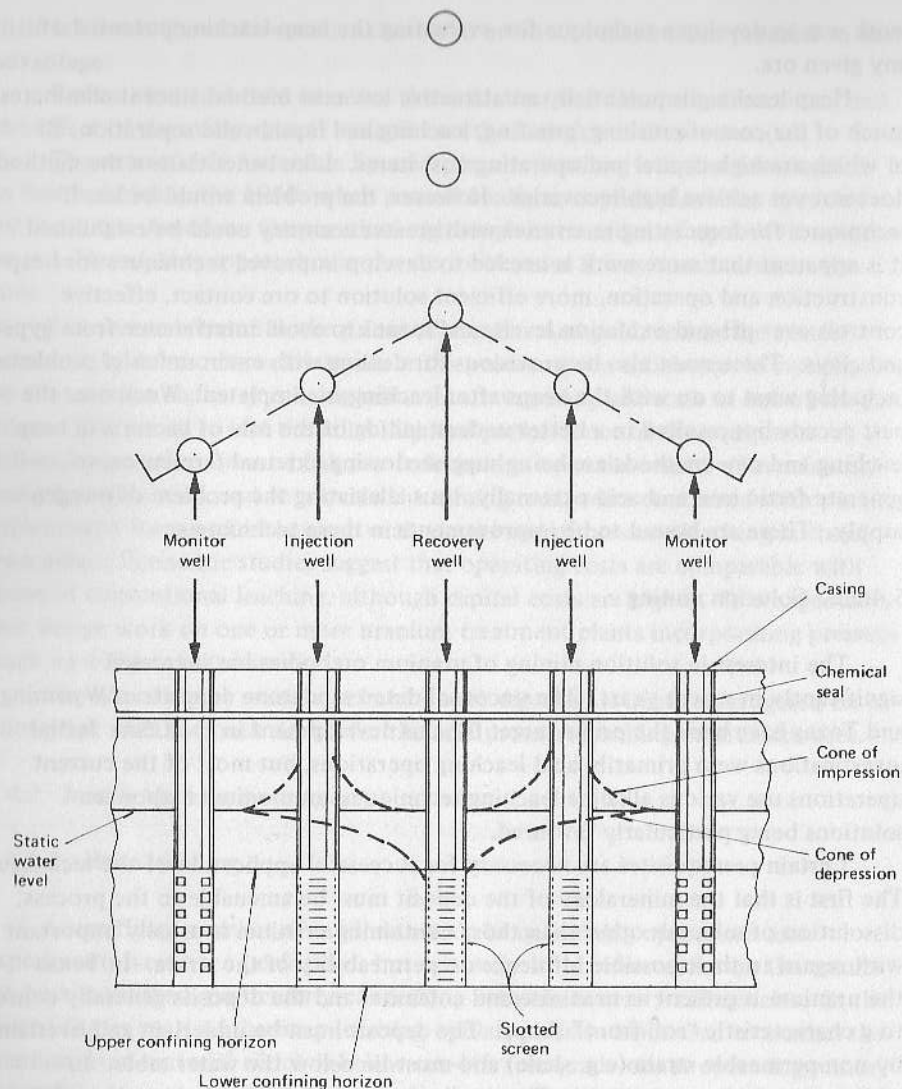


FIG.18. Solution mining for in-situ leaching of uranium.

where static water levels could be measured, to ensure that injection and recovery solution flows are balanced; in this way the body of leach solution can be contained with minimum environmental contamination.

The current arrangement now favoured is the five-spot pattern; injection wells are drilled in a 15-m-square pattern and are located at the corners of a square configuration with the recovery or production well in the centre. Monitor

wells are located at appropriate points to detect escape of leach liquor or ingress of groundwater. Well bore-holes are approximately 15 cm in dia. with casings of about 10 cm in dia. Reported recovery well flow-rates vary from 2 to 16 m³/h. Grades of pregnant solution vary from 10–20 to 200 ppm U₃O₈.

Assuming 3.3 m³/h output per well, 80 ppm U₃O₈, and 100% recovery of uranium on surface, a 113 400 kg U₃O₈ per year operation would require 71 production wells. To this must be added an equal number of injection wells, and a network of monitoring wells to detect solution "excursions".

In 1976 Westinghouse estimated the capital cost of a 131 m³/h, 113 400 kg U₃O₈ per year operation at \$US 2 000 000. The indicated solution grade for this operation approaches 150 ppm U₃O₈, and a total operating crew of 40 men may be required. Westinghouse stated that the total cost of the above operation (capital plus operating) would be in the order of \$US 55/kg U₃O₈. The major operating costs include chemicals, drilling and aquifer restoration.

One of the major problems of solution mining is loss of well permeability with time. Blockage of a well can occur owing to precipitation of insoluble compounds (e.g. CaCO₃), or swelling of clays which are normally present in significant concentrations. The choice and chemical makeup of the leach solution employed are major factors in controlling loss of well permeability. Most operators currently use an ammonium carbonate/bicarbonate leach solution along with hydrogen peroxide as an oxidant; however, the relative advantages of the ammonia-based system over a sodium-based system do not appear to be well defined. Leach solution pH is stated to be a major factor in maintenance of well permeability. It is also considered important to keep the concentrations of the various ionic species as close as possible to natural orebody conditions. To this end, some operators use groundwater from the aquifer in question for make-up leach solution.

The uranium recovery process first involves treatment of the uranium-bearing liquor in filters to remove any sand or other solids, followed by contact in ion-exchange columns. The barren solution from ion exchange is returned to the well for re-injection after chemical make-up. It is sometimes necessary to remove calcium from the barren solution by precipitation. The uranium is stripped from the resin by elution with sodium chloride solution. The pregnant eluate may be passed through charcoal columns to remove impurities, including molybdenum, and then to precipitation tanks where yellowcake is precipitated with ammonia or caustic soda.

Solution contaminants leached from the orebody often necessitate additional processing steps to avoid operating problems. For example, the chloride content of some leach solutions has been relatively high, with a correspondingly depressing effect on resin loading and production capacity. Considerable problems have also been encountered with scaling of pipelines on surface and underground owing to formation of calcium scale.

As a result of the requirement for control over solution concentrations, and also due to the fact that more solution must be withdrawn from the orebody than is pumped into it, there may be a significant bleed of solution from the circuit. Disposal of this excess solution may be via evaporation ponds, but in any event is subject to environmental restrictions. The degree of recovery obtainable from an "average" orebody is not known; no operator has attempted to obtain a reliable measurement on a well field or well pattern by re-drilling an exhausted area.

Although present well design and completion techniques appear to be generally satisfactory, experimentation continues with well-field design techniques. The deepest deposit at present in commercial operation is 80 m below surface. Owing to uncertainty of recovery estimates using solution mining techniques, it would be difficult to construct a good comparison with conventional mining/milling for a case where either would be economic. For deposits that are unattractive for conventional methods, the low capital investment required for in-situ leaching makes this approach worth consideration.

5.4.2.3. Underground in-situ leaching of broken ore

A novel in-situ leaching technique is being used at the new Agnew Lake operation in Canada (Lendrum and McCreedy 1977 [48]). Special underground mining techniques are used to blast and cave the ore into large stopes. Acidic leaching liquors are applied to the broken ore by spray pipes at the top of the stopes. The solution percolates through the broken ore and is collected in sumps. The collected liquor is pumped to the surface for circulation through the surface heap or for processing. The "swell" produced by the difference between the volume of broken and unbroken ore is taken to the surface for heap leaching. The operation is designed to take advantage of the catalytic action of thiobacillus ferrooxidans to utilize the air to oxidize iron and maintain a high redox.

5.4.3. Strong acid cure leaching

A process that is generating increasing attention for the leaching of certain types of uranium ores is the strong acid cure leach developed by the United Kingdom Atomic Energy Authority and applied continuously by Pechiney Ugine Kuhlmann.

Depending upon the system selected, crushed or ground ore is wetted and mixed (agglomerated) with concentrated (4 to 6 normal) sulphuric acid; the mixture is then cured. The reactions generate heat and the temperature increase assists solubilization of the uranium. After curing, an acidic spray wash is applied to the mixture. After water washing, the leached and drained solids are removed

mechanically and discarded to the disposal area, and the uranium-bearing solutions are then processed for recovery of yellowcake.

Details are given in Part II of the equipment and operating conditions of the strong acid cure leach as applied by Pechiney Ugine Kuhlmann at the Arlit plant, Somair in Niger.

As an example of the economics of strong acid leaching, the expected advantage in 1975 of the UKAEA process over a conventional 2720 t/d uranium mill, located in the Elliot Lake district, Canada, was \$2 096 000¹ in capital costs and the operating savings were estimated at \$0.37/t ore [49]. However, before the installation of the UKAEA process could be considered, a pilot-plant campaign was required. The 1975 capital costs of a 9 t/h and an 18 t/h pilot plant were estimated as \$600 000 and 800 000, respectively; operating costs were respectively \$3.85 and 2.31/t. The total cost of the pilot-plant study was estimated to be \$150 000 – 200 000.

At the Naturita, Colorado, operations of Ranchers Exploration and Development Corporation, uranium-vanadium tailings are now being retreated using a strong acid process. The reclaimed tailings are agglomerated with sulphuric acid, and placed in prepared earthen vats for subsequent counter-current percolation leaching. The collected solutions are processed through solvent extraction for recovery of uranium and vanadium. This is the only commercial operation in North America employing the strong acid leach; an ore treatment plant did operate in Wyoming for a short period in the 1960s using a strong acid-vat leach-ion exchange flowsheet.

5.5. Solids/liquid separation in high-rate thickeners

A high-rate thickener which has been designated the "Enviro-Clear" thickener, has been installed in the washing circuit of the Bear Creek mill of the Rocky Mt. Energy Co. in Wyoming. Figure 19 shows a generalized view of the unit. The flocculated feed slurry is introduced horizontally and at a controlled rate into a previously formed sludge bed rather than above the bed as in a conventional thickener.

The sludge bed provides a filtering effect on the liquid and the need for a large clarifying area is thereby eliminated. As a result, space requirements are reduced substantially. The Enviro-Clear equipment is being used for clarification and for yellowcake thickening, as well as in CCD.

Initial experience in North America with the Enviro-Clear in acid leach CCD circuits has been generally favourable, but there is insufficient experience to evaluate fully the merits of this equipment. There was some concern that the Enviro-Clear thickeners would require close operator attention and that a high

¹ Canadian dollars used in this section.

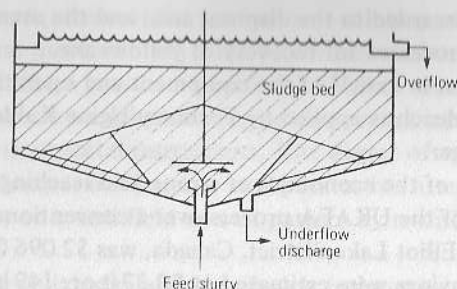


FIG.19. Enviro-clear thickener.

floculant dosage would need to be maintained; the former has not materialized to any significant extent.

Three 11-m Enviro-Clear thickeners have recently been included as sixth-stage vessels on three CCD circuits at an acid leach uranium mill in Elliot Lake, Canada. Also at this location, a 9-m Enviro-Clear vessel is being used to clarify acidic mine waters after neutralization.

In South Africa, a 17-m-dia. Enviro-Clear thickener is being tested to thicken old tailings recovered from a pond area. It is reported that a 17-m and 24-m dia. unit will be installed in separate mining locations in South Africa.

In North America, both Dorr Oliver Inc. and the Eimco Division of Envirotech Corporation are pilot-plant testing thickeners, of a novel design, which are expected to have high capacities similar to that achieved with the Enviro-Clear design.

The development of deep-well floc-blanket thickeners will no doubt continue and it is possible that a combination of the deep-well feed to handle the bulk of solids with the lanellar thickener to handle the overflow slimes may maintain the competitiveness of the CCD washing system so far as capital cost is concerned.

5.6. Concentration/purification by CIX

The application of small manually operated fixed-bed ion-exchange plants (FBIX) at small mines remains attractive. (The keynote of its simplicity has already been highly exploited for water treatment.)

However, the disadvantages of FBIX become ever more evident as the scale of operations increases. On the other hand, with the inevitable trend to lower-grade solutions, solvent extraction (SX) also becomes less attractive. For these reasons the development of continuous ion-exchange (CIX) equipment that can overcome the difficulties encountered in traditional fixed-bed installations,

and the use of resin ion exchange for the concentration and purification of pregnant liquors, have probably received more attention in recent years than any other part of the uranium ore processing flowsheet. The three most important of these defects of FBIX are as follows:

(i) The size to which a fixed-bed unit can be increased without fall-off in performance is limited. This size limitation leads to the necessity for a large number of parallel trains of ion-exchange columns when a high flow-rate of solution is to be treated. No savings resulting from large-scale operation are possible since not only ion-exchange vessels but also pumps, valves, piping, control equipment, operational manpower, maintenance and analytical services are all increased in direct proportion to the number of units required.

(ii) Poor use is made of the exchange power of the resin. At any time in the exchange cycle there is always an appreciable fraction of the total resin inventory that is not exchanging ions. While countercurrent contact of resin and solution flows is readily achieved in CIX designs, this is not so for FBIX. In fixed beds the bottom of the resin bed is not eluted as well as the top. When pregnant solution flows through this bed, fed in the same direction as the eluate, the phenomenon of self-elution is frequently observed. Here traces of uranium remaining at the bottom of the bed are eluted by the barren solution passing through this region. This uranium is not recovered.

(iii) Fixed-bed units require a clarified solution since a resin bed acts as an efficient filter for any slime present, resulting in hydraulic problems and eventual blinding of the bed.

The development of CIX technology has not been without its pitfalls. A review paper published in 1972 [50] gave illustrations of 27 different CIX systems and described many others. Brief descriptions are given below of four of the CIX systems that have been included in industrial uranium plants.

5.6.1. The CHEM-SEPS system

The CHEM-SEPS continuous ion-exchange system (also known as the Higgins loop) is the oldest of the CIX techniques that have been applied to uranium recovery. Although pilot-plant test work was conducted in 1955 and a number of water treatment plants are in service, it is only recently that this system has been specified for a commercial uranium plant established by Wyoming Mineral (Higgins, 1978 [51]) Corporation at Bingham Canyon, near Salt Lake City, Utah, USA. The plant was supplied by Chemical Separations Corporation and is used for recovering uranium from a copper-tailings leach solution that contains only 10 ppm U_3O_8 . The CHEM-SEPS CIX system is shown in Fig.20. It is a single loop comprising four sections which are separated by valves to isolate the resin undergoing different treatment stages in the ion-exchange cycle. The resin is

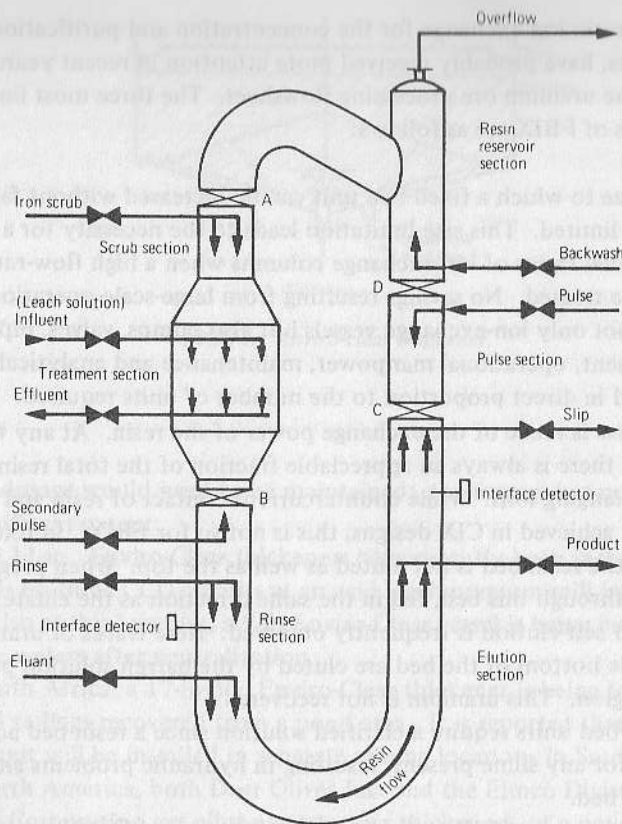


FIG.20. CHEM-SEPS uranium recovery system.

moved hydraulically as a compact bed in small increments. During the period when the bed is stationary, the solutions flow countercurrent to the direction of resin movement, and more than one liquid stream may enter or leave a given section. Rinsing solution and fresh eluant enter the lower section, while loaded eluant and displaced pulses of liquid are removed at points further round the loop.

Probes that detect changes in concentration control the position of the interfaces between these different solution flows. Provision is made for the backwashing of resin after it has been loaded but before it enters the pulse and elution sections.

It is claimed that the CHEM-SEPS CIX system can handle feeds containing substantial amounts of suspended solids. However, the ability of the system to clear itself of suspended solids that have become trapped by the resin bed in the loading section is a function of the resin circulation rate. For the situation in

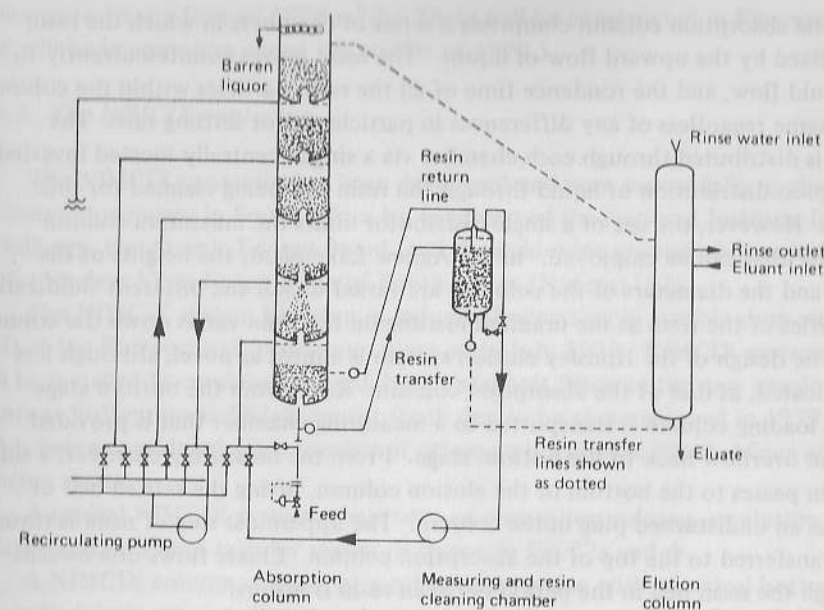


FIG.21. Himsley CIX system.

which the uranium concentration of the feed is relatively low and the suspended solids content high, the danger exists that the rate of resin and suspended solids to the backwash chamber will not equal the rate of accumulation of trapped solids in the loading section. Although external timers are used to order the major periods in the exchange cycle, considerable dependence is placed on the internal probes necessary for interface control.

A major advantage claimed for the CHEM-SEPS system is the relatively high flow-rates per unit of cross-section that are possible. As a result this system has been used for applications where the concentration of the ion to be extracted is very low, as is the case for tailings solutions and water-treatment applications.

5.6.2. The Himsley system

A Himsley CIX system has been included in the uranium plant of Kerr Addison at Agnew Lake, Ontario, Canada. A typical Himsley unit is shown in Fig.21. It comprises an absorption column, a resin-measuring chamber, an elution column and a rinsing chamber. In this system there is no interruption of the feed liquor when resin is transferred from the absorption column, each batch of resin being measured and moved separately and consecutively down the column.

The absorption column comprises a series of chambers in which the resin is fluidized by the upward flow of liquor. The resin moves countercurrently to the liquid flow, and the residence time of all the resin particles within the column is the same regardless of any differences in particle size or settling rate. The liquid is distributed through each chamber via a single, centrally located inverted weir, good distribution of liquid through the resin bed being claimed for this device. However, the use of a single distributor limits the maximum column diameter that can be employed. In the Agnew Lake plant, the heights of the stages and the diameters of the columns are varied to suit the different fluidization properties of the resin as the uranium loading on the resin varies down the column.

The design of the Himsley elution system is almost as novel, although less complicated, as that of the absorption column. Resin from the bottom stage of the loading column is transported to a measuring chamber that is provided with an overflow back to the bottom stage. From the measuring chamber, a slug of resin passes to the bottom of the elution column, lifting the settled bed of resin as an undisturbed plug in the column. The uppermost slug of resin is rinsed and transferred to the top of the absorption column. Eluate flows downwards through the resin bed in the period between resin transfers.

The Himsley contactor installed at Agnew Lake is the first plant of this design in commercial operation. The requirement that the plant should be erected in an existing building led to the division of each of the two absorption columns into two sections in series, each 7.3 m high and 2 m in dia., containing five and four compartments, respectively.

During start-up there was a high concentration of nitrate in the feed, probably due to some excessive residue from the explosives used in the initial blasting of the Agnew Lake ore. This resulted in nitrate levels in the feed of 2 g/litre. At the same time, the molar ratio of ferric iron to uranium approached 30:1, which, together with the nitrate, reduced the loading of uranium and slowed down the rate of absorption. This necessitated operation at low flow-rates to obtain acceptable barren effluents.

More recently, a new section of the mine has produced liquors of a reducing nature, resulting in about 100 ppm of polythionates appearing in the pregnant feed. This fouled the resin to the extent of 140 g/litre or more, but subsequent treatment with caustic soda and sodium chloride regenerated the resin. It has been reported that 316 stainless steel is severely corroded in such a reducing environment, particularly in the elution column, where the level of polythionate is high.

Despite these unexpected problems, the mill staff has kept the system in operation, producing acceptable barren solutions. At an operating temperature of 15°C, each column is treating about 50 m³/h.

(A Himsley contactor is being designed for Vaal Reefs Exploration and Mining Company Limited in South Africa, comprising five 4-m dia. absorption

columns to treat a flow of 1050 m³/h. These will be constructed in Fiberglass and will be in operation about the middle of 1979.)

5.6.3. The NIMCIX contactor

The NIMCIX contactor has been developed and very successfully applied to uranium recovery in South Africa by engineers of the National Institute for Metallurgy, the Atomic Energy Board, and the gold-mine groups who are linked to the Nuclear Fuels Corporation of South Africa (Nufcor) [52].

The NIMCIX system has been in industrial operation in combination with CCD at the Blyvooruitzicht uranium plant since July 1977. NIMCIX systems will be installed in combination with horizontal belt filters in the new uranium plants at Stilfontein and Kleinespruit (both due to be commissioned in 1979) and is being considered for a number of other applications in South Africa and in other countries.

A typical NIMCIX system comprising an absorption column, an elution column and two resin transfer vessels, is shown in Fig.22a and b.

A NIMCIX column consists of a cylindrical section with a conical bottom. The cylindrical section is flared at the top, above the top surface of the fluidized bed of ion-exchange resin. The long cylindrical section is divided into a number of contacting stages by distribution trays that allow countercurrent and periodic movement of the solution and the resin. The feed solution enters through a distribution pipe below the bottom tray and the barren solution leaves the top of the column via a peripheral overflow into an overflow launder. The overflow launder can be fitted either inside or outside the main shell of the column. Resin is added to the top of the column through a central feed-well which extends below the surface of the liquid to just above the top surface of the fluidized bed of resin. Resin is withdrawn through the apex of the bottom conical section. The bottom tray is modified by the addition of caps placed over each perforation. This combination has been designed to achieve a self-regulation of resin inventory throughout the column.

The operating cycle of the NIMCIX contactors consist of the following five distinct periods:

The forward-flow period: Solution flows upwards through the column and fluidizes the resin in each stage. (In the absorption column, the resin beds expand to the height of each stage.) The only wasted volume is the space just below each tray in which the solution rearranges itself and flows through the various orifices in the distribution tray.

The settling period: During this period, the solution flow is stopped and the resin settles in a dense state on to each distribution tray. This ensures that

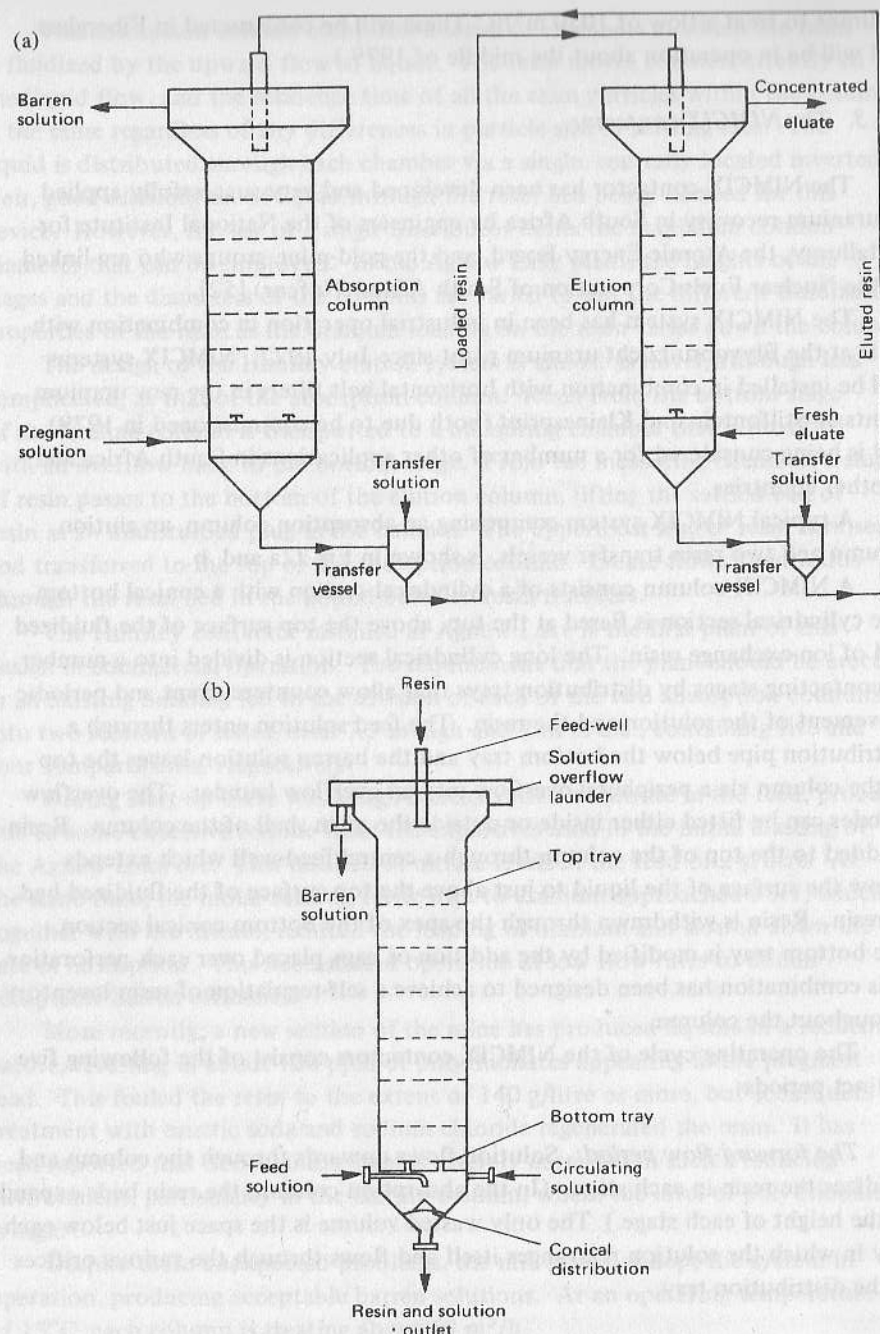


FIG.22. (a) and (b). The NIMCIX recovery module.

the resin is transported in a dense state during the next period in the cycle and the back-flow of solution between stages is minimized. The caps on the bottom tray of the column ensure that no resin drops into the conical section of the column during this period of the cycle.

The reverse-flow period: A known volume of solution is withdrawn from the column through the transfer vessel by the circulating pump. Simultaneously, a known volume of resin is transferred between the stages and also out of the bottom stage. At the end of the reverse-flow period the resin that has left the bottom stage is distributed between the conical section of the column, the pipe connecting the bottom of the column with the transfer vessel, and the transfer vessel itself.

The delay period: Solution is circulated through the conical section of the column and the transfer vessel by the circulating pump. This ensures that all the resin in the conical section of the column and in the line from the column to the transfer vessel is deposited in the transfer vessel. The caps on the bottom tray of the column again ensure that no resin drops into the conical section of the column during this period of the cycle.

The resin-transfer period: The transfer vessel is isolated from the column during this period. Solution is introduced into the transfer vessel by the transfer pump and the resin is transferred to the top of the elution contactor.

The forward-flow and resin-transfer periods start simultaneously. The steps described ensure that the resin movement is stable and predictable, that no valves close on the resin, and that resin movement is gentle.

The elution contactor is generally smaller than the absorption contactor and thus has to complete more than one cycle to every cycle of the absorption contactor to preserve the resin balance. Finer control of the resin balance is achieved by the use of an ultrasonic level-detector mounted in the top of the elution column. During the first settling period following a resin-transfer period, the detector senses whether the level of the resin in the top stage of the elution column is too high or too low. If the level is too low, a long forward-flow period timer is selected. If the level is too high, a short period timer is selected. The level of the resin is thus maintained between two desired limits.

Eluted resin is transported periodically to the top of the absorption column. The expanded cross-sectional area ensures that the introduction of resin creates little disturbance of the upward flow pattern. The possibility of entrainment of resin in the barren solution overflow is thus minimized. An additional allowance in the height of the uppermost stage provides space for the most recently transferred slug of resin without the necessity for the prior removal of the existing stage inventory.

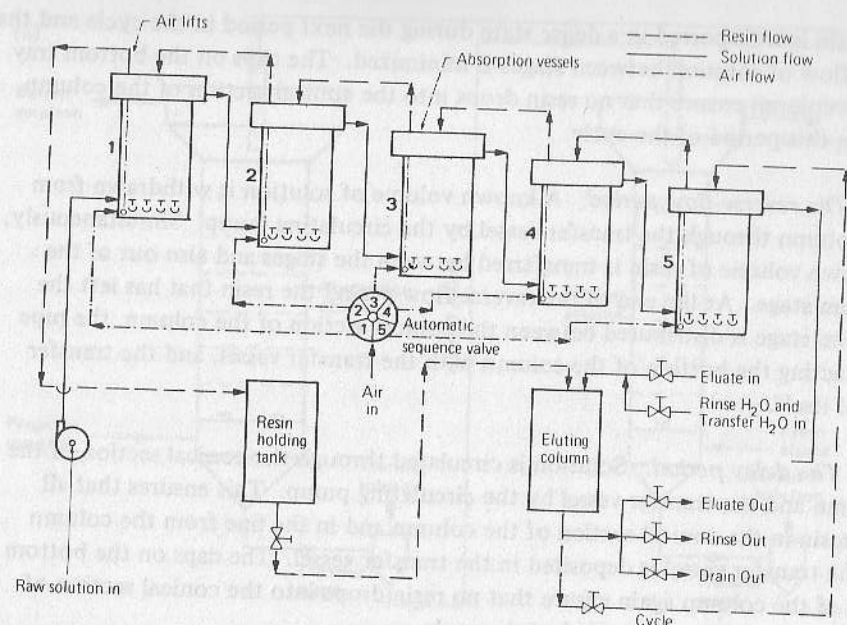


FIG.23. Porter ion-exchange system.

5.6.4. The Porter system

The Porter CIX design concept, as specified in a US Patent of 1975, has been applied commercially at Rössing and in two solution mining projects in Texas.

In the Porter CIX system, shown in Fig.23, countercurrent resin-solution contact is made in a series of vessels. In each vessel the solution is introduced through a branched manifold, flows upwards through a fluidized-resin bed, and overflows into a launder before being pumped to the next stage. Resin is moved countercurrent to the solution flow by airlifts operating from the bottom of each CIX contactor to a raised vessel from which the resin gravitates to a still-well in the centre of the next stage. Resin movements are a series of batch operations. Resin is more or less completely transported from the first vessel of the series before the resin contents of the second vessel are transferred to the first vessel, and so on down the line to the last vessel, into which eluted resin is introduced. The solution flow is uninterrupted, but only four of the five compartments can be counted as active stages at any one time. A certain amount of dead volume in each compartment is unavoidable in this design. Any resin that is entrained in the liquid overflowing from a stage will be subject to passage through a centrifugal pump. As in the screen-mix RIP systems operating in the USA,

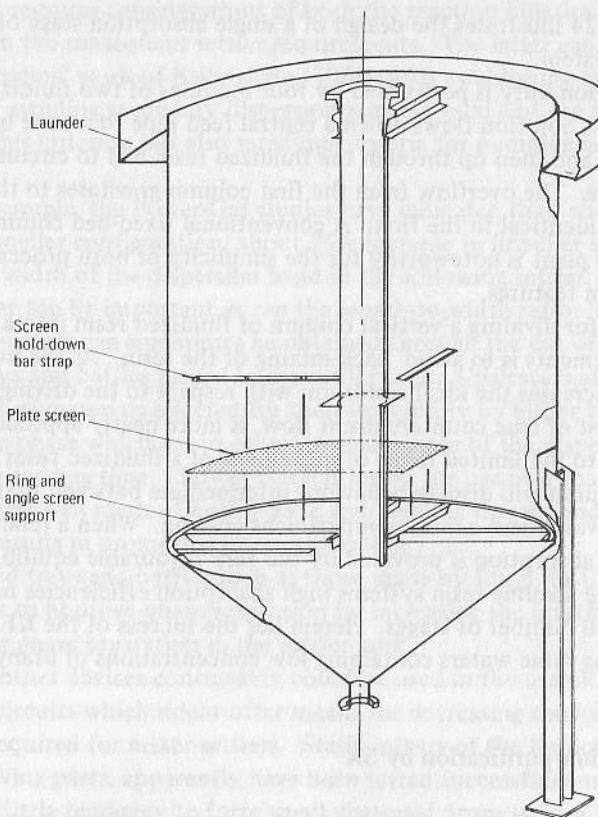


FIG.24. KERMAC CIX system (one absorption stage).

problems may arise in the maintenance of an even distribution of the resin inventory between stages. The CIX plant at Rössing consists of four absorption modules in parallel each comprising six rectangular Porter contactors. The elution section comprises three conventional fixed-bed ion-exchange columns per module.

A second Porter CIX plant has recently been installed at Rössing to treat a recycled stream of low uranium concentration. The absorption compartments in this plant are circular in cross-section and interstage solution flows by gravity.

5.6.5. The KERMAC CIX system

Keer-McGee Corporation have been recovering uranium from mine waters for more than 15 years in a CIX plant designed by their own staff specifically for the

purpose. Figure 24 illustrates the design of a single absorption stage of the KERMAC CIX system.

The absorption duty is performed by four modules of two fluidized-bed columns in series. Solution flows down a central feed pipe to a cone below a perforated plate, and then up through the fluidized resin bed to circumferential overflow launders. The overflow from the first column gravitates to the second column which is identical to the first. A conventional fixed-bed column is used for elution. This plant is noteworthy for the simplicity of both process and mechanical design features.

The reason for dividing a vertical column of fluidized resin into a number of separate compartments is to avoid back-mixing of the resin. As the number of compartments increases, the ideal condition with respect to the driving force for mass transfer, that of true countercurrent flow, is more nearly approached. However, owing to the limited range of movement of a fluidized resin particle, an undivided column will display behaviour intermediate between that of a perfectly mixed vessel and a multicompartment column. When a relatively large driving force for absorption is provided by the very favourable equilibrium that is shown in dilute alkaline resin systems high absorption efficiencies may be realized in a small number of stages. Herein lies the success of the KERMAC system in treating mine waters containing low concentrations of uranyl carbonate.

5.7. Concentration/purification by SX

All mills at present using solvent extraction are operating mixer-settler units and treating clarified pregnant solutions. Perhaps the biggest drawback to these circuits are the large quantities of solvent required to fill the system. Other problems usually are associated with emulsions and cruds which interfere with operation and which often are difficult to overcome. Not enough is known about the influence of solution impurities, chemistry, dynamics etc., in uranium solvent extraction from leach liquors, and the industry could benefit if comprehensive studies were undertaken to develop better understanding of these systems. Although the solvents now in use do their job well, it also is not unreasonable to believe that better solvents could be developed which might have more selectivity, lower aqueous solubility and less tendency to form the cruds and stable emulsions. More work probably should be done on the carrier solvents that are in use since some investigators have indicated that the extraction kinetics and disengagement rate may be improved with the proper choice of diluent as compared to the conventional kerosene. Mixer-settler design probably has not been perfected to the point that truly optimum performance is achieved, although a considerable amount of recent work has been done by Wellman-Power Gas Inc. and others to improve the design. Impeller design seems to be quite

critical and requires considerations of both the reaction kinetics and the effect of mixing in the subsequent settler requirements. The latter can be quite important.

The original work of Bellingham (1960 [40]) establishing criteria for adequate mixing but avoiding secondary dispersion remains valid and the problem of satisfying this criterion but also satisfying criteria for pumping is not yet well resolved.

For example, the Power-Gas studies have indicated that, with constant flow-rate and impeller configuration, about 25% increase in impeller speed will nearly double the width of the dispersion band in the following settler. Flow distribution in the settler can be important as can the length-to-width ratio. Improvement in area requirements can sometimes be obtained through the use of coalescers, or stacked trays and baffles placed in the settler tank. Coalescers are devices constructed of materials selected for their wettability by either the solvent or the aqueous phase which act to promote coalescence of the dispersed phases and thus reduce settling time. Power-Gas has tested the use of a coalescer consisting of a kit interwoven mesh of both plastic and metallic threads and has obtained promising results in promoting phase separation.

Stacked trays and baffles such as those made by Lurgi, IMI, Pielkenrood, and Heil act to improve phase separation by increasing the interfacial areas available for phase separation in the settler tank.

Many other devices conceivably could be used in the uranium solvent extraction circuits which might offer means for decreasing the large solvent inventory required for mixer-settlers. Static mixers of the Kenics type, which have no moving parts, apparently have been tested successfully and probably would have little tendency to form small dispersed drops or lead to entrainment. Phase separation, therefore, should be rapid and settler size smaller than usual. Nothing seems to have been published about the use of this type of mixer with regard to the kinetics of the exchange reaction or the transfer obtained.

Ritcey has recently presented a comprehensive evaluation of a number of different types of solvent extraction device with potential application in the uranium industry [53]. Landau et al. [54] have reported specifically on developments in the use of reciprocating-plate extraction columns for hydrometallurgical application. Other than the mixer-settler, probably the most interesting of these are the pulsed columns, primarily because of their relative simplicity and their potential use in solvent-in-pulp (SIP) systems. Solvent inventory would be less for the columns and less floor space would be required, although the columns would require more head room. A deterrent to the use of these devices is the difficulty of scaling up test data to commercial size and the lack of industrial experience in their use. However, pulse columns have been in use in the uranium refineries for many years. Column scale-up is difficult since true staging is not obtained as in mixer-settlers, and it is necessary to consider many additional factors in translating from a small test unit to commercial-size devices.

In spite of some projected problems, the pulse column does present some interesting possibilities and deserves more study. At present, the main research in this area is being done in Canada by Ritcey and his associates. Of particular interest has been their work on the application of the sieve-plate pulse column to direct extraction of uranium from leach pulps. Considerable potential is indicated for this method in that liquid-solid separation (and the associated costs) can be completely eliminated without impairing the overall metallurgy. A principal problem has been the solvent loss; however, this can apparently be controlled even at the present level of achievement to only about two and one-half times the usual losses encountered with typical SX practice in the USA, which has been approximately 1 litre mixed solvent per cubic metre feed solution, or roughly 0.5 litres/t of ore. Pulse columns up to 25 cm in dia. have been operated in a pilot plant; however, further studies undoubtedly will be needed before scale-up to commercial size is attempted.

5.8. Precipitation

Of all the unit operations included in conventional uranium circuits, information on precipitation phase has received least publicity. An Australian paper appeared in 1968 [55] but other operators appear to have kept their experiences unpublished. In many plants, conditions are such that precipitation is routine and so long as conditions are properly controlled little attention is required to obtain a satisfactory product. In other plants, however, the problem of producing high product purity, high density, and good filtration and washing characteristics may require considerable attention. In general, the precipitation technique remains almost totally an art, affected by such unpredictables as system geometry, agitation, continuous versus batch systems, methods of reagent addition and timing, and other factors.

Some fundamental studies on uranium precipitates have been undertaken in recent years in an effort to increase our knowledge of the precipitation process. At least three laboratories have undertaken a systematic study of ammonium uranate (yellowcake) precipitation products to elucidate the composition of different solid phases found when ammonia is added to a solution containing uranyl ions.

In another area of investigation, increased attention has been given to use of precipitants other than ammonia. A number of uranium mills are conducting laboratory and plant tests on the use of hydrogen peroxide for uranium precipitation. The peroxide process has been adopted by the Atlas Minerals Uranium Mill in Moab, Utah, for use in selectively precipitating uranium from sodium chloride strip solutions containing relatively high concentrations of vanadium. Two other mills are currently using hydrogen peroxide on an intermittent basis: Union Carbide Corporation in Uravan, Colorado, and Rio Algom

Corporation in La Sal, Utah. Users of hydrogen peroxide report they obtain a product of greater purity which is more readily filtered, washed and dried than comparable ammonia-precipitated uranates.

Quite obviously there is a need for in-depth study of other factors influencing the type of uranium compounds precipitated from solutions, with the goal of achieving better control of the reactions, improvements in the process techniques, and high-grade products with optimum physical characteristics.

5.9. Flowsheet innovation

Should uranium processing be reduced to its minimum elements, we would find these to be size reduction, leaching, purification and precipitation. Even these steps might be simplified and the purification step eliminated if more specific leaching techniques or more specific precipitation techniques could be found to isolate the uranium from other elements present. This is unlikely in the leaching step if the lixiviant is sufficiently reactive to achieve high extraction, although alkaline leaching with carbonate comes close.

Both the RIP and SIP processes provide the means whereby the circuit can be reduced to these minimum elements since, given the proper resins, solvents and equipment, it seems within our reach to eliminate liquid-solid separation and classification steps from the present flowsheets. This, by the way, provides another cost advantage over the associated capital and operating costs for these operations, since we also avoid the loss of soluble uranium which would normally be present in the washed tailings from either thickeners or classifiers.

It is also possible that some of the above steps could be consolidated, again with the right equipment and material. For example, a coarse resin with sufficient abrasion resistance might be added directly to the leach circuit. With a counter-current flow of resin and pulp and with proper staging, such as is now done with slime pulps in several RIP circuits in the USA, the end-products could be barren leached pulp and loaded resin ready for elution.

Some studies have been done years ago to achieve direct precipitation of the uranium product from a solvent, thus combining what would normally have been the solvent stripping and precipitation operations. These tests were not particularly successful since precipitation was incomplete and too much solvent was lost. This remains an interesting concept, however, and a study in sufficient depth could be rewarding.

Looking to the longer-term future, and in particular at the problems of exploiting uranium deposits that will require expensive deep level mining for their exploitation, an interesting new development is in progress in South Africa.

The Chamber of Mines Research Organization has made considerable progress developing a metallurgical process which could be employed underground, in conjunction with compact milling machinery.

The first stage in the process is to crush and grind the ore underground. For this the Research Organization, in association with an international process company, is developing a mill which occupies only a fraction of the space required by conventional milling machinery. It has been found that it is not necessary, at this point to grind the ore to the fineness required to release completely the gold and the uranium.

The metallurgical concentration process carried out underground is divided into two phases. In the first, minerals such as pyrite and thucolite (containing significant quantities of gold and uranium) and any free gold released from the rock, are concentrated in two steps, firstly by flotation and secondly by the application of hydrocyclones.

According to the Research Organization, more than 98% gold, 90% uranium and 99% pyrite can, typically, be concentrated into 30–40% by mass of the rock fed. This means that there is a substantial reduction in the volume of rock which has to be hauled to the surface for treatment.

The waste pulp which is drawn off at the top of the hydrocyclone is largely free of valuable minerals and is considered to be an almost ideal material to fill those underground areas which have been worked out. Because the flotation process effectively removes most of the fine particles, the waste material is relatively coarse and, as a result, is free-draining. Once placed in the worked-out stope as pulp the water drains away leaving 60 to 70% of the stope volume filled with rock.

Although a number of engineering problems still have to be overcome it is hoped to have a demonstration plant operating underground at a scale of 100 t/h by 1982.

Advantages claimed for this exciting new South African development are:

- (i) The relief of limitations on hoisting capacity, possibly permitting a higher rate of mining in the area serviced by a single shaft;
- (ii) Improved strata control because of the extensive use of backfill, with a consequent improvement in productivity and also the possible extraction of remnants which are currently left on reef, particularly in deep-level mines;
- (iii) Improved ventilation control because of the filling of worked-out areas;
- (iv) A dramatic reduction in the amount of timber required, and a consequent reduction in the risk of fires;
- (v) Revised layouts in mines to facilitate the movement of men and materials, with a consequent improvement in both productivity and communication in the mine; and
- (vi) A doubling of the grade of material received from the mine, making it possible for most gold mines to become uranium producers (a reverse leach would be desirable to overcome the effects of the flotation reagents in the concentrate).

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PART II

SELECTED EXAMPLES OF SPECIFIC URANIUM DEPOSITS AND OPERATIONS

CASE STUDIES ILLUSTRATING THE CHARACTERISTICS OF THE VARIOUS ORE TYPES

The uranium deposits have been classified for this report, according to the scheme proposed by Ruzicka [1]. Although this classification was drawn up for geological purposes and is based on ore genesis hypotheses, it is convenient to maintain a nomenclature that has become relatively common usage for summarizing ore processing characteristics.

Figure 25 is adapted from Ruzicka and shows the distribution of uranium resources according to ore type.

1. SANDSTONE DEPOSITS

Epigenetic uranium mineralization in fluvial, deltaic and other alluvial sediments forms the genetic classification of sandstone-type ores. They should not be confused with sandstones or similar sediments containing syngenetically deposited detrital uranium minerals which can be as different from the aspect of processing as from that of ore genesis. A substantial fraction of the world's uranium derives from this ore type located in the USA. Niger and Argentina also produce uranium concentrates from sandstone ore and deposits are known to occur in South Africa, the Indian subcontinent, Europe and Australia.

The mineralogy and lithology of the host rocks generally conforms to their origin, and ranges from quartz conglomerates to fine siltstones, with the sandstone facies predominating. The matrix between the main quartz grain body contains clay minerals in varying amounts together with calcite and iron oxides. Induration varies from loose to friable, lightly cemented sand grains to quartzite but, in general, the sandstones exhibit a fairly high degree of porosity. However, although this porosity has effected a significant degree of stratigraphical control, the deposit may not be confined to a specific strata so that a particular ore lens may include a variety of lithologies, all to be mined and fed to the mill.

Carbonaceous material from plant remains and of "kerogen"-like character are commonly but not ubiquitously associated with uranium mineralization. Pyrite is generally present and ferroselite can also occur.

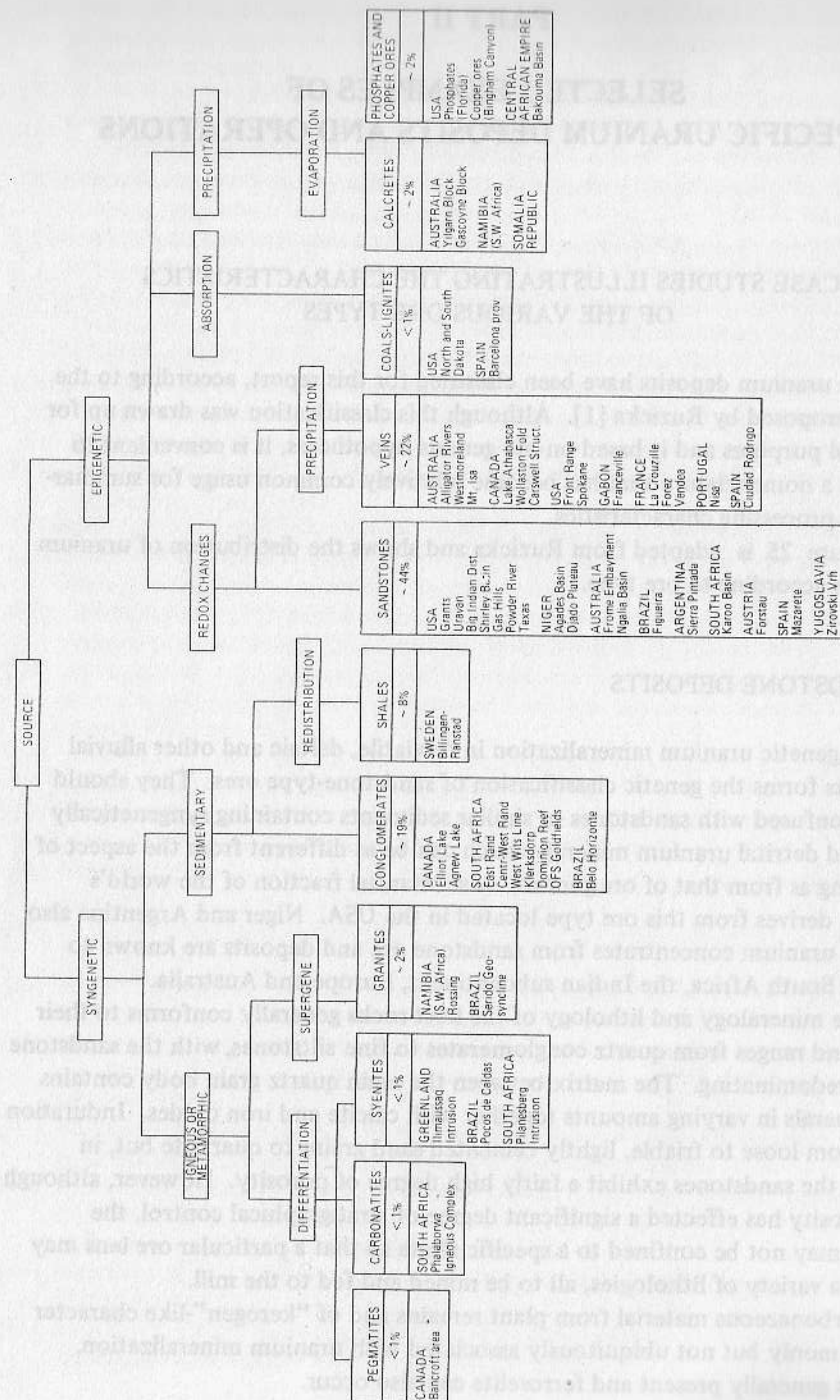


FIG. 25. Distribution of uranium resources (RAR + EAR below US\$50/lb U_3O_8). (Statistical data derived from *Uranium Resources, Production and Demand*, OECD-NEA/IAEA Dec. 1977. Classification scheme from Ruzicka [5]). RAR = Reasonably assured resources; EAR = Estimated additional resources.

The mode of mineralization results in suites of minerals typifying their origin from aqueous solutions, their deposition being controlled by pH and Eh conditions and their composition by solubility criteria pertaining to the elemental composition of the solutions.

From the aspect of ore processing this form of ore genesis has produced minerals from which it is not generally difficult to take uranium into solution. Uraninite and coffinite predominate in the unoxidized ores. Where aqueous oxidation has taken place hexavalent uranium silicates and oxides are found, but frequently the uranium is then fixed with vanadate and phosphate to form minerals such as tuyamunite, carnotite and autunite. However, some uranium in these ores is also usually associated with organic material and with clay minerals from which form it is more difficult to extract.

1.1. United States of America

To the end of 1977 approximately 221 000 t uranium had been produced in the USA essentially from sandstone-type deposits, which still contain about one third of the world uranium resources (excluding the central economy countries). There were at one time over 950 producing mines, many quite small and only a few large operations. By 1977 this declined to 180 mines with the ore being processed in 16 mills, with several more planned. There are three main areas: the Colorado plateau, the Wyoming basins and the Texas coastal plain. These deposits are classified into three types as follows:

Peneconcordant: "Trend" ores occurring in the Mesozoic sediments of the Uruan and Grants mineral belts of the Colorado Plateau; they are epigenetic deposits rounded to rectangular in shape comprising tabular ore bodies parallel to the paleochannels.

Stack deposits: Redistributed ore occurring also in the Mesozoic sediments of the Colorado plateau, particularly in the Ambrosia Lake region of the Grants mineral belt. Epigenetic mineralization similar to peneconcordant but controlled partly by fault zones giving rise to lenticular ladder like deposits.

Roll-front deposits: Also epigenetic and in origin similar to the other two; they form typically crescent-shaped roll deposits with mineralization following the contact between altered and unaltered sandstones. They occur particularly in the Tertiary sediments of the Wyoming basins and Texas with some also in the Colorado plateau.

1.1.1. Grants Mineral Belt — Colorado Plateau

Most of the uranium in the Grants Mineral Belt occurs in the Westwater Canyon Member of the Morrison Formation. The rest occurs in the Jackpile

sandstone of the Bushy Basin Member of the Morrison Formation. During 1953 to 1959 ore from the Todilto limestone was also treated by an alkaline leach plant operated by Anaconda.

Because of the great extent of the areas involved, the number of mines and the variations inherent in rocks, in mill feeds are exhibited all the extremes in physical and chemical mineralogy from the sediments. In 1977, for example, Anaconda received ore from 12 sources, Kerr McGee from 37, United Nuclear Homestake from 25, and Sohio from 19.

Plants currently in operation are Anaconda-Bluewater (6000 t/d), Kerr McGee (6500 t/d), United Nuclear-Homestake (3500 t/d), Sohio-Cebollete (1500 t/d) and United Nuclear-Churchrock (2700 t/d). The discovery rate of deep, unoxidized, trend ore has been significant during the past few years with major deposits being found in the Churchrock, Crownpoint, east Ambrosia Lake, and Laguna areas.

Plants in the planning stage are Gulf-Mount Taylor (4500 t/d), Phillips Petroleum-Nose Rock (3000 t/d) and Mobil Oil-Crown Point (3000 t/d). Drilling is continuing down dip from the main ore trends in an attempt to follow the Churchrock and Crownpoint trends eastward towards Mt. Taylor. Further north, drilling in the vicinity of Nose Rock continues in the 1000–1300 m depth range. Exxon has commenced exploration on its Navajo lease in the western San Juan Basin.

(a) Mineralogy

Oxidized and unoxidized ore is mined, with the latter predominating. In the more typical sandstones there are two distinct types usually associated with pre-fault or post-fault conditions. The first is grey, black or brown and owes its colour to black organic matter which has been shown to have properties similar to “kerogen”. This material tends to coat sand grains, cover clay particles and fill interstices.

The second ore type has a purplish, brown colour varying to dark grey and almost white. It contains less organic matter although when coloured the sand grains carry a thin film of brown material which includes coffinite. Vanadium minerals are frequently present in this ore both as individual minerals and as vanadium bearing micas and clays.

The mineralogy of the host rocks is typical of such sedimentary materials; quartz pebbles and sand grains, partly or rarely wholly cemented with quartz or carbonate. Calcite occurs to a varying degree with montmorillonite clays and chlorite. Although stratigraphical controls tend to confine the mineralization to the cleaner sandstones there is sufficient mudstone and calcareous material to provide a significant tonnage of ores with high clay or high carbonate contents. These call for care in blending for mill feed and such variations have played as big a role as uranium mineralogy in affecting flowsheet variations. Of specific

minerals present coffinite, urano-organic complexes and uraninite are the most common uranium minerals in unoxidized ores. Tyuyamunite, carnotite and uranophane occur in oxidized ores. Pitchblende occurs in deep unoxidized deposits. Of minor importance in oxidized ore near the surface are andersonite, zippeite, bayleyite, autunite and meta-autunite. Apart from the main gangue minerals vanadium, molybdenum and selenium occur as accessory minerals but not generally in economically recoverable amounts. Associated minerals are paramonstroeseite, pyrite, jordisite, ferroselite, and marcasite. The calcite content varies from 2 to 5% and, as indicated above, varying amounts of dolomitic and bentonitic clays occur. Mining ore grade varies from 0.05 to 0.25% U_3O_8 with average grade to the mills of approximately 0.15% U_3O_8 .

(b) Comminution

In contrast to the fine grind required for conglomerates, sandstone ores require only that the natural grain size be liberated and the uraniferous matrix material be moderately comminuted to expose the uranium minerals to the action of leaching reagents.

These characteristics have resulted in a trend towards simple open-circuit milling with high pulp densities in rod mills and semi-autogenous mills to achieve separation of and some scrubbing of sand grains but to minimize grinding energy.

However, Mesozoic rocks of Colorado plateau are generally more indurated than the Tertiary sandstones of Wyoming and Texas. Kerr McGee uses open-circuit rod mills, Homestake mill uses ball mills (in presence of recycled carbonate solutions) while the Anaconda, Sohio and Churchrock plants have adopted semi-autogenous milling. The acid leach plants grind to 80% minus 35 mesh while United Nuclear-Homestake grinds to 80% minus 100 mesh.

(c) Preconcentration or beneficiation

Sandstone ores consist essentially of two components — fine to coarse clastic material and a cementing matrix which contains the uranium. The fine dispersed nature of the uranium mineralization precludes the successful use of physical concentration processes. Some success has been obtained with the use of flotation to separate the feed ore into high and low calcite fractions or high and low sulphide fractions. In these instances, however, both streams still have to be leached for uranium. However, this approach has not been adopted on an industrial scale in the Grants Mineral Belt.

For the friable sandstones, ideally an abrasive, scrubbing type of grind might produce a coarse sand fraction which would be sufficiently barren to reject. This has not been achieved in general practice. Simple screening beneficiation has been used by small mines to reduce tonnage for transportation to the mill. However,

such treatment effects little saving in leaching reagents because the carbonate and other reactive gangue reports also on the fine fraction so that the value of uranium lost in the sand fraction is not compensated for and no advantage is gained at the mill itself.

(d) Leaching

United Nuclear Homestake use alkaline-carbonate leaching (either atmospheric or pressure) for treating a wide variety of ores from the Grants Mineral Belt. When silicate-type uranium minerals are present, carbonate leaching is accomplished with difficulty. Oxidized uranium minerals are readily soluble in carbonate solutions but tetravalent minerals require intensive treatment with oxidation and long residence time at high temperature and pressure. All the other plants use acid leaching which is more effective with difficult ores, requiring less grinding of the ore, less leaching time and lower temperatures and providing more flexibility to deal with changing ore characteristics than alkaline leaching.

Original feed can vary from 1.0 to 9.0% CaCO_3 and blending is used to achieve an average CaCO_3 to the mill for economic acid leaching of between 2.0 and 4.5% in various mills. This results in acid consumption of the order of 40 to 60 kg/t ore.

Heating with steam is common practice to achieve leaching temperatures of between 40 and 60°C depending on conditions. This particularly accelerates the extraction of uranium from organic material and clays.

Roasting in an oxidizing atmosphere improves uranium extraction from Jackpile sandstone ores. The porosity of the ores is improved by the removal of carbonaceous materials, by dehydrating clays and by breaking down agglomerates of a clay-silicate-organic nature. The result is better access of the leaching agent to the uranium minerals. Recovery varies from mill to mill and also appears to depend on grade being around 95% for 0.20% and 93% for 0.15% ore.

(e) Solid-liquid separation

Homestake use three stages of filtration in rotary-drum filters. Kerr McGee carry out a sand/slimes separation in hydrocyclones. Sands are washed in five-stage rake classifiers while slimes are washed in six-stage CCD in conventional thickeners. Anaconda have recently introduced a seven-stage CCD circuit based on Enviroclear hi-rate thickeners. It would therefore appear that the presence of clay minerals in the ores of the Grants Mineral Belt make solid-liquid separation in both alkaline and acid circuits relatively difficult.

(f) Concentration purification

The presence of molybdenum and vanadium in the ores gives rise to a certain amount of difficulty in achieving satisfactory product purity. Homestake have neither ion-exchange nor solvent extraction after leaching and have had to introduce roasting and water leaching of ammonium diuranate to reduce vanadium to specification limits. Kerr McGee have to increase the organic-to-aqueous ratio in the solvent extraction plant or raise the amine content when the Mo content of their feed ore is high. Part of the organic phase is treated in a separate Mo stripping circuit to minimize molybdenum-amine sludge formation.

1.1.2. Wyoming Basin

Uranium occurs in coarse arkosic unconsolidated sandstones of the Wind River Formation (Shirley Basin and Gas Hills), the Wasatch Formation (Powder River Basin) and the Battle Spring Formation (northern Great Divide Basin). Significant deposits have been discovered in recent years in the Fort Union Formation (southern Powder River Basin). The highest grade uranium ores and most of the resources are in roll-type deposits which are tongue-shaped in plan and crescent-like in vertical section with uranium mineralization following the contact between oxidized and unoxidized sandstone. In 1977 Federal American treated ore from 9 sources, Lucky Me-Gas Hills (9), Union Carbide (6), Western Nuclear (5), Lucky Me-Shirley Basin (1) and Exxon (2).

Plants currently operating area: *Gas Hills area* — Federal American (900 t/d), Lucky Me (1500 t/d), Union Carbide (1100 t/d) and Western Nuclear (1500 t/d); *Shirley Basin* — Lucky Me (1600 t/d), Getty Oil (1600 t/d), Powder River Basin-Exxon (2700 t/d) and Rocky Mountain Energy (900 t/d). Recently activities have centred on deep underground mining, solution mining, and the planning of new ore processing facilities in the central and southern part of the Powder River Basin where at least six deposits have been discovered. Kerr McGee plan to start up an 1800-t/d plant while United Nuclear/TVA are considering building a 2000-t/d mill to treat several orebodies in the southern area. It is possible that two or three additional plants will eventually be established in the area but timing is uncertain.

(a) Mineralogy

The mineralogy of the deposits also shows wide extremes in matrix composition, particularly as far as clay and silt content is concerned. The minerals in the ore zone occur in a fairly characteristic pattern across the width of the roll from the unaltered sandstone to the altered sandstone.

The most important uranium minerals in the unoxidized zones are uraninite, pitchblende and coffinite, while uranium phosphates, silicates and hydrous oxides are found in the oxidized zones. In the Gas Hills area uranium generally occurs as earthy brown to black coatings and interstitial fillings of uraninite and coffinite. Autunite, meta-autunite, phosphuranylite and metatyuyamunite are the most common secondary minerals in the oxidized zone. Over 20 minor uranium minerals have been identified. In the Powder River Basin, coffinite occurs as thin sooty layers and spherical masses on detrital sandstone grains.

Selenium is enriched on the convex side of the roll fronts as ferroselite, while molybdenum sulphide and calcite are enriched on the concave side. Some deposits contain bentonitic clays. Gypsum and pyrite occur in varying amounts.

At Shirley Basin pyrite is abundant in the ore zone. In some places, it replaces or is disseminated in carbonized fossil remains. Associated minerals are calcite, hematite and marcasite.

Calcite occurs as cementing material but its distribution is mainly in relatively discrete concretionary masses within less calcareous rock.

Phosphate is low and, although vanadium has been recovered from Shirley Basin ore, in general it is low compared to the Colorado plateau.

Ore grades are generally in the range 0.10 to 0.15% U_3O_8 .

(b) Comminution

Although induration is less than the Colorado plateau ores, liberation characteristics and degree of grind for Wyoming sandstone ores are similar to that described above for the Grants Mineral Belt. Federal American use ball mills, Lucky Me-Gas Hills, Union Carbide-Gas Hills and Exxon-Powder River use open-circuit rod mills, while the other plants have adopted semi-autogenous milling. The bentonitic clay minerals can cause handling problems with wet or frozen ore, in Wyoming, in terms of blending, storing and milling. Gas-fired rotary kilns are used at some plants for the drying of ores.

(c) Preconcentration or beneficiation

Physical separation characteristics and amenability to preconcentration are similar to that at Grants (above). None of the plants carry out preconcentration.

(d) Leaching

All the plants use acid leaching with sodium chlorate as oxidant and temperatures in the range 35–55°C. Lucky Me-Shirley Basin uses air-agitated pachucas while all the other plants employ mechanical agitators. Attempts have been made to treat 0.03% U_3O_8 ore by in-situ and heap leaching. However, liquor

recovery from in-situ leaching is very poor and heap-leach piles tend to seal off with consequent low percolation rates.

(e) Solid-liquid separation

Federal American, Union Carbide and Western Nuclear opted for sand/slime separation followed by resin-in-pulp recovery of uranium from dilute pulps (up to 15% solids). The other plants either use CCD in thickeners directly on the leached pulp or for slimes washing after sand/slime separation in cyclones. The Rocky Mountain Energy-Bear Creek mill, commissioned late in 1977, was the first plant to install Enviroclear thickeners for CCD.

(f) Concentration/purification

Either solvent extraction or a combination of ion-exchange and solvent extraction (Eluex) is used for uranium recovery. No particular problems are experienced in achieving the desired product purity.

1.1.3. Texas coastal plain

Uranium occurs mainly in the Stone's Switch and Tordilla Sand Members of the Whitsett Formation (Karnes and Live Oak Counties) and at the Palangana Salt Dome (Duval County).

Lenticular orebodies in both occur in elongated clusters. Grades range from 0.05% U_3O_8 to as high as 0.5% U_3O_8 , the latter being uncommon. Crescent-shaped roll-front alteration interfaces are present in classic form in many deposits. In others, oxidation has apparently distorted or dispersed upper portions of the interface so that only less permeable siltstones, claystones and lignites retain a significant thickness of uranium mineralization.

Solution mining operations currently in production are US Steel-Clay West, US Steel-Burns Ranch, Union Carbide-Palangana, Intercontinental-Pawnee and Wyoming Minerals (Bruni). Two new solution mining plants will be in production during 1978. One conventional mine/mill is in operation (Conoco-Falls City – 2000 t/d) with another under construction (Chevron-Panna Menia – 1800 t/d). In 1977, Conoco treated ore from 14 different sources.

(a) Mineralogy

The Stone's Switch ores are fine-grained, silty, limey sandstones containing shallow oxidized deposits while the Tordilla ores are dirty, poorly-cemented, fine- to coarse-grained sandstones containing unoxidized bodies.

Uranium occurs in the tetravalent state, i.e. as coffinite or uraninite in unoxidized deposits and as uranium phosphates, silicates and hydrous oxides in oxidized ores.

Autunite, carnotite and tyuyamunite are the most common uranium minerals in the near surface ores mined by open-pit methods.

Relatively large amounts of bentonitic clay occur in the ore mined by Conoco-Falls City and in deposits being exploited by solution mining. Precipitation of uranium is thought to have been caused by reduction with H_2S and carbonaceous material is rare in south Texas sediments except in lignite beds (Jackson Group) and lignitic silts (Catahoule Tuff). Ores in the Conquista-Falls City area are generally lower in carbonate content and more amenable to acid leaching than deposits to the southwest. Small amounts of disseminated pyrite and marcasite occur.

(b) Comminution

Comminution is not applicable for solution mining plants. Comminution at Conoco-Falls City is by crushing and open-circuit rod milling with provision for roasting of wet ores.

(c) Preconcentration or beneficiation

This is not applicable for solution mining plants; and not considered by Conoco or Chevron for the same reasons that rule out preconcentration of other US sandstone ores.

(d) Solid-liquid separation

Agitation or solid-liquid separation equipment is not required for solution mining plants — Conoco leach in mechanical agitators and carry out solid-liquid separation in a five-stage CCD thickener circuit.

(e) Leaching and uranium recovery

The question of containment of leach liquors within the well field periphery was a major obstacle in solution (borehole) mining, but engineering developments now appear to have overcome the problem to a large extent. Continuing operational control of fluids can be maintained through an effective method of well layout, construction and maintenance. One of the latest innovations are well patterns aligned to cover mineralized outlines. This offers the possibility of mining small, sinuous ore zones which otherwise might not be exploitable.

In many of the known uranium-bearing formations that are being exploited by solution mining, a substantial part of the matrix is comprised of calcium-

bearing clays (e.g. sinectite). One of the most effective lixiviants for uranium extraction from such formations is an aqueous solution of ammonium carbonate and/or bicarbonate. However, ammonium ions in the lixiviant exchange with calcium ions in the clays, are retained by the clay, and bleed into the underground water thus causing contamination. Also, exchanged calcium ions react with carbonate lixiviant to form insoluble calcium carbonate (calcite) which can cause serious plugging throughout the leach circuit. A novel technique has been developed by one company which overcomes both these problems to a large extent. The lixiviant used has a cationic composition essentially the same as that of the underground formation water, with only small concentrations of dissolved CO_2 and O_2 (as an oxidant) therein. The pH of the lixiviant injected into the well is maintained at about 7.5 to 8.0 (i.e. at the same value as the underground formation water) by lime addition, in a calcium precipitator. After calcium removal, suspended solids are removed from the leach solution by filtration and the uranium recovered by ion-exchange. Barren solutions from ion-exchange are passed to a mixing tank where CO_2 is added to make up fresh lixiviant for injection, together with additional oxidant, into the boreholes. No cations (NH_4^+ or Na^+) are added to the lixiviant so that there is substantially no contamination of underground waters. Also, since the cationic composition of the lixiviant is essentially the same as that of the underground formation, there is little, if any, swelling of clays which can decrease the permeability of the orebodies. Finally, plugging due to underground precipitation of calcite is avoided. Recovery of uranium is by fixed-bed ion-exchange at US Steel-Clay West, while continuous ion-exchange is used at US Steel-Burns Ranch (Porter system) and at Union Carbide-Palangana (modified USBM MCIX column). Conoco uses solvent extraction.

1.2. Niger — Agades basin

Uranium occurs in the Agades basin on the western sides of the Air mountain range. The geological setting of the deposits is a delta system cut by a complex network of fluvial channels. The ores consist basically of alternate layers of sandstone and clay. The sandstones consist primarily of silica clastics and are relatively porous. In the case of the Arlit deposit 70–80% of the uranium occurs in the sandstones and 20–30% in the clays. The deposit which supplies ore to the operation at Arlit has reserves of 30 000 t uranium in ore of average grade 0.25% U_3O_8 .

This deposit has been mined and treated to produce yellow cake since 1971. The Somair Arlit plant was initially designed to produce approximately 750 t U/a as sodium diuranate (approx. 70% U_3O_8 grade). Capacity has been increased in stages to approx. 1500 t in 1977 and a total of approximately 7700 t has been produced.

A second plant is now being commissioned (Cominak-Akouta) while two more plants are under consideration (Arni and Imouraren).

The Niger government encourages a diversity of partners in developing the uranium industry and is engaged in joint exploration ventures with a number of foreign companies, including Conoco, Urangesellschaft, PNC-Japan, Esso, CEA-France, Pan Ocean-Canada, Agip-Italy. While prospecting operations are continuing in the Agades basin, large areas are also now being investigated in the extreme north of the country (Emi-Lulu and Djado regions).

(a) Mineralogy

The Arlit uranium deposit mined by open pit is a mineralized zone in argillaceous sandstones, sandstones and micro-conglomerates. The sandstone is the main facies and the cementation matrix is mainly kaolinite or dickite. Pyrite is common as sand coatings or intergranular masses. Black, argillaceous masses occur and the mineralization is associated with carbonaceous matter. Some baryte occurs and also calcite in both sandstone and clays. However, carbonates are of minor importance.

Uranium in the sandstones at Arlit occurs as pitchblende and coffinite in very rich samples. In medium-grade sandstones (0.2% U_3O_8) uranium is contained mostly in the kaolinite or dickite cementing matrix material, without any definite mineralization with the exception of secondary carnotite. In the clays it was impossible to identify any differentiated uraniferous mineralization — the uranium is entirely bound to the clayey phylitic phase together with organic matter. The clays contain substantial quantities of acid-consuming constituents such as Fe_2O_3 , Al_2O_3 , CaO , MgO , K_2O and reducing organic materials. The sandstones contain less acid-consuming materials and very little organic matter. Both types of ore contain molybdenum and vanadium.

(b) Comminution

Uranium is contained in the fine matrix material in the case of the sandstones whereas it is distributed throughout the particle-size range in the case of the clays. At Arlit the ore is first crushed to minus 300 mm and then milled to a nominal grind of 600 μm (at which point the sandstone grains are liberated) using dry autogenous grinding in a cascade-type installation (drying, pneumatic classification).

(c) Preconcentration or beneficiation

Techniques such as attrition scrubbing, gravity concentration, flotation etc., were tried on the Arlit ore but were found not to be applicable. Firstly, grinding was unable to achieve complete separation of the uraniferous kaolinite matrix

from the barren silica grains. Thus a discardable tail could not be produced. Secondly, the clay material did not respond to physical methods of concentration and the uranium concentration remained constant between all size fractions of clay material.

(d) Leaching

The treatment process is the original application of a semi-dry strong acid process.

Conventional acid leaching, whilst providing 97% dissolution of uranium in the 50–100 μm size range failed to produce more than 80% uranium extraction in the minus 50 μm range. This latter material, however, responded to strong acid concentration (600 kg H_2SO_4 /t ore) to provide an overall extraction of 96%. Strong acid leaching, in order to be economic, requires a much reduced liquid to solids ratio. In the case of the Arlit plant the ore, ground to 600 μm , is preheated in a Louisville tube, then impregnated with 50% H_2SO_4 in a pelletizing drum, cured in a second drum for 3 h and then ground in a pebble mill before repulping with water to extract the solubilized uranium. The Cominak-Akouta plant uses a similar strong acid leach process but curing is on a conveyor belt instead of a rotating drum.

(e) Solid-liquid separation

The sandstone and clay components have different rheological properties in aqueous suspension. At Arlit rake classifiers are used for prior removal of the sand fraction to avoid overloading of the 6-stage CCD thickeners, owing to fluctuations in the sandstone: clay ratio in the feed. Five-stage rake classifiers are used for sands washing. Solid-liquid separation at Akouta will include the use of three 60 m^2 area, horizontal belt, vacuum filters. Belt filters are also being considered for Imouraren.

(f) Concentration/purification

The higher molybdenum content of the ore to be processed by Cominak-Akouta requires a more complex solvent extraction process than that used by Somair-Arlit. The leach liquors contain 5–6 g/litre U_3O_8 and 1–3 g/litre Mo. The solvent extraction circuit comprises extraction, selective uranium stripping using a sodium chloride solution, molybdenum stripping using sodium carbonate (which is also the means of solvent regeneration), and a final solvent washing with water.

1.3. Argentina — Sierra Pintada

Sandstone-type uranium deposits of economic size and grade have been found in several provinces of Argentina. Several small operations have produced approximately 25 to 30 t U/a since the late 1960s, the total production to end of 1977 being approximately 450 t U as yellowcake. The largest deposits are in the Mendoza province where mineralization is in Cretaceous and Permian sediments of typically sandstone, conglomerate, siltstone facies. A small mill has operated for ten years at Malarque at a throughput of approximately 100 tons/d with ore grade of 0.15 to 0.20% U_3O_8 . Development exploration has proved reserves of 10 000 to 15 000 t uranium oxide in a grade of 0.10 to 0.15% and a new mill with a nominal capacity of 600 t U_3O_8 /a is being installed.

(a) Mineralogy

Most of the mineralized bodies in the Sierra Pintada district of Mendoza province are located in fine- to coarse-grained sandstones of abundant quartz and lithic fragments, well graded, and yellowish to greenish in colour. The rocks and ore are relatively highly indurated, with siliceous and calcareous cementation material. Calcium carbonate is relatively high, between 7 and 8%, and clay minerals make up 12 to 15%. Mineralization is as coatings and intergranular streaks and masses. Uranium occurs at depth as uraninite with thucolite and other asphaltic materials. Pyrite is associated with uranium together with other sulphides, notably chalcopyrite and chalcocite. In the oxidized zone uranium occurs as uranophane, carnotite, tyuyamunite and lesser amounts of other secondary minerals, and pyrite has gone to iron oxides and copper to carbonates.

(b) Comminution

The circuit for comminution is relatively conventional with two-stage crushing and closed-circuit rod milling to a nominal size of 100% passing 700 μ m.

(c) Preconcentration or beneficiation

In common with other sandstones, no method is applied at Malarque. However, a deposit at Cosquin contains 20 to 25% calcium carbonate and both gravity and flotation were examined. Grade was increased from the range 0.03 to 0.08% U_3O_8 up to approximately 0.14% with up to 75% recovery and 70% carbonate rejection. Calcite in the concentrate was still considered too high for economic leaching with acid.

(d) Leaching

Acid leaching is used with acidity at 0.9 to 1.0 pH and temperature 50°C. The combination of high carbonate and significant uranium in carbonaceous material results in high acid consumption (approx. 140 kg/t) and low recovery (approx. 90%). Calcination was examined but not considered to achieve sufficient improvement for the extra cost.

A small orebody at Don Otto Near Salta has been treated by heap leaching. The following conditions were used:

Heaps were prepared in fixed sites, i.e. the ore was placed for leaching and removed after leaching. The normal leach pile contained 4000 t run-of-mine ore at approximately 65 mm top size. Four piles were in operation each 20 m \times 36 m \times 3.5 m deep, laid over asphaltic paper, protected with approximately 25 cm gravel. The leaching cycle consisted of:

- 10 days "soaking"; solution of fresh acid pH 0.8,
approximately 140 litres/t
- 150 days leaching; recycle solution pH 1.2, 40 litres/t
- Total acid consumption was 15 kg/t and recovery 50 to 60%.

The ore handled well on to the piles but broke down during leaching causing significant blinding.

(e) Solid-liquid separation

The plant at Malarque uses one stage of primary thickening and two stages of filtration.

(f) Concentration/purification

Conventional amine solvent extraction is used and no particular problems are encountered.

At Don Otto the liquor was originally precipitated with lime and the crude concentrate despatched to Malargue. However, a small fixed-bed ion-exchange plant has been installed to produce yellowcake.

2. QUARTZ-PEBBLE CONGLOMERATES

Uranium-bearing, pyritic, quartz-pebble conglomerates similar in lithological and general mineralogy have been found in Canada, South Africa, Australia and Brazil. These deposits are restricted in geological time to the period following the development of an extensive acid crust, but preceding the development of an

oxidizing atmosphere. The uranium host rock generally consists of quartz pebbles in a quartzite matrix with pyrite and minor silicates. The uranium mineralization which is associated with the fine matrix is mostly found in the older (Proterozoic) strata.

Two of these occurrences — in Canada and South Africa — have provided and will continue to provide a significant fraction of the world's uranium supply. The South African deposits were among the first low-grade ores commercially exploited, coming into production at the beginning of the 1950s. The Canadian deposits have been extensively worked from the mid-1950s. Geologically they are placed under a common classification which for the exploration geologist and mining engineer is a most valid and useful classification.

Indeed, also the general mineralogy is such that they are both treated using the acid leach process. However, the detailed uranium mineralogy is significantly different so that the conditions for leaching also are different.

2.1. Canada: Elliot Lake — Agnew Lake District

Ore deposits occur at the base of the Matinends Formation on the limbs of a large folded structure which consists of the Quirke Lake syncline and the Chiblow anticline. A major feature is the remarkable uniformity and continuity of the ore-bearing rocks over large areas. The Elliot Lake deposits are not as deep as the Witwatersrand deposits and their uranium grade is six times higher.

During the late 1950s there were eleven mills processing between 30 000 and 35 000 t/d of this general type of ore with a feed grade between 0.10 and 0.15% U_3O_8 .

Expansion plans by Denison and Rio Algom will result in a considerable increase in tonnage treated in the Elliot Lake District. Denison plans to expand their 6400 t/d mill to around 10 900 t/d by 1982, the additional ore coming from reactivation of the Stanrock mine. A subsequent expansion is possible to increase the plant to about 13 600 t/d by 1985. Rio Algom are currently expanding their mill to about 6350 t/d. A second phase expansion will include reactivation of the Panel mill (3000 t/d, starting up 1979) and the Preston Mines — Stanleigh mill (4500 t/d, start-up 1983/84).

(a) Mineralogy

Typically the host rock is a quartz-pebble conglomerate with generally well-rounded pebbles ranging from 150 mm to gravel size. The rock is solidified by a hard crystalline quartzitic matrix. In addition there are some silicates such as chlorite, sericite, and feldspar. The pebble to matrix is approximately 2:1. Mineralization is mainly within the matrix and consists of pyrite with minor amounts of apatite, ilmenite, zircon, rutile and leucoxene. The principle uranium

mineral is brannerite with lesser uraninite and minor amounts of uranothorite thucolite, coffinite and gummite. Monazite is relatively plentiful.

(b) Comminution

Because of the quartzite matrix breakage occurs across the pebbles so that, unlike sandstone deposits, uraniferous conglomerates require grinding of both the barren pebbles and the uraniferous matrix material, and separation of the pebbles from the matrix for selective grinding of the latter is not possible. All the existing mills in the Elliot Lake area use crushing and two-stage grinding in closed circuit with classification (rod and pebble mills). The size of the solids fed to the leach is generally 80% minus 85 mesh (50% minus 200 mesh). Autogenous mills (with pebble ports) are being considered for Stanleigh.

(c) Preconcentration or beneficiation

For pebble milling about 5% of the ore is scalped out for media at -82.5 mm + 50 mm. Power consumption in pebble mills is about 14 kWh/t.

The amenability of a large number of preconcentration techniques was investigated by Rio Algom who found that only radiometric sorting, heavy media separation and flotation offered any potential at all. Only coarse ore could be radiometrically sorted and even then a relatively low ratio of concentration was achieved. The application of flotation and heavy media separation selectively recovered acid-consuming minerals with the uranium and a discardable tailing could not be produced. None of the plants in the Elliot Lake area uses preconcentration or radiometric sorting.

(d) Leaching

Typical conditions for leaching to achieve 90 to 95% recovery are summarized as follows:

The neutral slurry is thickened and filtered to achieve a slurry concentration of between 70 and 75% by weight of solids (S.G. of ore is 2.7).

Acid addition: Approximately 35 kg H_2SO_4 /t ore to maintain approximately 1N free acid.

Temperature: Steam is injected to achieve 70 to 75°C.

Time: Minimum 40 h but can be up to 96 h.

Oxidant: The quantity varies from nil to a little. Sodium chlorate has been used.

The main feature is the relatively aggressive condition required in the leaching process. In this feature the Blind River ores could be typified as brannerite ore (rather than "conglomerate" ore). The high free acidity required throughout the leaching time, the temperature and to some extent the time of leaching are characteristic of the conditions required to open up brannerite and uranorthorite.

However, the actual consumption of acid by the ore itself during the leaching operation is only about 17 to 20 kg/t ore reflecting the good fortune to have few reactive gangue minerals associated with the brannerite. Dyke material (and the underlying chlorite schist) are to be avoided since their inclusion in the mill feed results in excessive acid consumption in the leach.

By leaching at as high a concentration of solids as possible the remainder of the acid, required in order to maintain the acid concentration at approximately 1N, can be kept to a minimum (also here about 17 to 20 kg/t ore). These circumstances mean that there is little economic margin available for beneficiation processes, unless they achieved dramatic improvements in grade with very little loss of uranium.

In an effort to utilize the excess remaining acid, two-stage leaching has been used. However, there is debate as to whether any real savings in cost are achieved when the extra operating procedures are accounted for, particularly when leaching can be done using such high concentrations of solids in the slurry (70 to 75%).

The presence of a substantial amount of pyrite indicates a good prospect for bacterial and "autogenous" leaching. Both techniques have been investigated for application to these ores. High-temperature, pressure leaching with oxygen was demonstrated in the laboratory and could have significant advantages, particularly if a relatively good supply of acid from other sources was not available.

Bacterial assisted in-situ leaching of broken ore in underground stopes is now being carried out on a large scale at Kerr Addison's Agnew Lake mine. This approach has proved feasible because the orebody is confined between impervious rock layers, dips 70 degrees from the horizontal and is highly fractured with all the uranium mineralization restricted to the fracture surfaces.

(e) Solid-liquid separation

Denison employ a sand/slimes split using hydrocyclones. Slimes are washed in a 3-stage CCD circuit using conventional thickeners. Sands are washed by a 2-stage rotary-drum vacuum filtration circuit. Rio Algom-Quirke have a 4-stage cyclone sand relief circuit-washed sands join washed slimes from a 6-stage CCD circuit using Enviroclear thickeners, at the neutralization pachucas prior to tailings disposal.

(f) Concentration/purification

Fixed-bed ion-exchange on clarified leach liquors is used by Denison, Rio Algom-Quirke and Rio Algom Panel. Continuous ion-exchange is used by Kerr Addison-Agnew Lake and is being considered by Stanleigh.

There are significant quantities of thorium and lanthanons in the liquors. These elements are associated with the uranium minerals in a number of minor heavy minerals and are at least in part taken into solution by the relatively aggressive leaching conditions. Thus, the following analyses are reproduced from Uranium in Canada [2]:

	Typical liquor analyses (g/litre)					
	Acid free	U ₃ O ₈	Th	SO ₄	NO ₃	Rare-earth oxides
Leach discharge	41	4.61	0.97	103.2	1.08	0.89
Pregnant liquor	0.9	1.02	0.14	18.1	0.79	0.18
Barren liquor	0.7	0.0011	0.15	18.9	1.24	0.23

These elements can cause some problems in the subsequent treatment by being extracted in small but significant proportions with the uranium and causing specifications to be exceeded for these elements. However, there is also an opportunity for economic by-product recovery which was practised for a time by at least two Canadian plants.

Other less significant features, but characteristic of the mineralogy, is the presence of polythionates and "ionic" silica in solution as a result of the attack on the sulphides and minor soluble silicates. These caused resin poisoning problems - believed to be aggravated when chloride elution was used.

2.2. South Africa: Witwatersrand. Orange Free State Goldfields

Important concentrations of uranium minerals occur in four continuous Precambrian systems, namely the Dominion Reef, Witwatersrand, Ventersdorp and Transvaal, of which the two first named are important because of their relatively large uranium content. The uranium grade is generally about one sixth of that of the Elliot Lake conglomerates and the deposits are emplaced at much greater depth.

The structure of the Witwatersrand System is that of an elongated basin stretching 320 km in a north-eastern direction and 160 km in a north-western direction. Most of the economically mineable gold and uranium occurs in conglomerate bands known as reefs or bankets in the Upper Division of the WWR system. In ascending order of depth the principal uranium-bearing conglomerates in the Upper Division are the Carbon Leader, belonging to the Main Reef Group, the White, Monarch, and one or two other conglomerate reefs of the Bird Reef Group on the West Rand, their equivalents, the Vaal Reef, in the Klerksdorp goldfield, the Basal and Leader Reefs in the Orange Free State goldfield, and the May Reef and its equivalents in the Kimberley Reef Group on the Far East Rand. The only uranium produced so far from the Lower Division of the WWR System has come from the Afrikaner and Inner Basin Reefs in the Government Reef Series west of Klerksdorp.

The Dominion Reef System comprises a basal group of conglomeratic, arkosic and quartzitic rocks up to 100 m thick, covered by about 900 m of lava of andesitic or acidic composition. The group of sedimentary rocks forming the base is, from an economic point of view, the most important of the Dominion Reef System. Two conglomerate bands along or close to the base of the sediment, which have been worked or prospected for gold and uranium, are known as the Upper and Lower Reefs. The Lower Reef has been the main gold carrier in the area. The Upper Reef is a small-pebble conglomerate passing laterally into a thin layer of grit containing scattered pebbles. Where it is thin the concentration of uraninite may be relatively high, with sections 2 to 5 cm thick.

Before uranium had any economic value, the conglomerates of the Witwatersrand had been worked for decades for their gold and the feed to the early uranium mills had originally been mined only for gold. Gold has remained the principal control on the mining operation and although the market place will always influence the situation and the uranium producers will no doubt tend to maximize the gross worth of the ore, the uranium grade has generally remained incidental so far as mining control is concerned. Mill-feed grades are low and generally around 0.02 to 0.03% U_3O_8 , ranging from 0.01% to rarely as high as 0.06%.

Up to the end of 1976 almost 75 000 t uranium metal equivalent had been produced.

Pyrite is also recovered for its value as the source of sulphur for acid production.

Uranium plants currently in production are Ergo (East Rand), West Rand Consolidated, Randfontein Millsite, and Randfontein Cooke (Central and near West Rand), Blyvooruitzicht, West Driefontein, and Western Deep Levels (West Wits Line), Fuffelsfontein, Hartebeestfontein, Vaal Reefs East, and Vaal Reefs West (Klerksdorp Area); Joint-Metallurgical Scheme-President Brand, Harmony and Virginia (O.F.S. Goldfields). New plants under construction or being planned are Vaal Reefs South, Stilfontein, and Merriespruit. Evaluation of the uraniferous

conglomerates in the basal sedimentary rocks of the Dominion Reef Group has indicated large deposits of uranium varying in grade from 250 to 1250 ppm U_3O_8 , but associated gold values are low and no decisions have to date been taken to exploit these deposits.

(a) Mineralogy

These deposits comprise a sequence of clastic sediments interlayered with volcanics. Enclosed in the clastic sediments are conglomerates comprising arkose and quartzite interbedded with silt and slightly metamorphosed.

In parts the conglomerates are thick and robust being similar in appearance to those of Elliot Lake, whereas elsewhere the conglomerate bands give way to thin beds of grit.

Uraninite occurs in the reefs as minute crystals, usually less than 0.08 mm across, and shows signs of rounding. The degree of roundness varies from angular to well-rounded. The tendency is for uraninite in reefs high up in the Witwatersrand succession to have a higher degree of roundness. In many samples the individual uraninite grains have been altered to such an extent by corrosion during metamorphism that little can be seen of their original shapes. Secondary uraninite sometimes encloses and replaces primary uraninite grains, and is in many places a prominent constituent of the Bird Reefs in the West Rand and Klerksdorp areas. Almost all uraninite grains contain minute spots and veinlets of galena in which the lead is mainly of radiogenic origin.

A variety of pebbles occur in the conglomerate reefs. Those of vein-quartz predominate and may be glassy white, mottled grey, black or opalescent blue in colour. Pebbles of quartzite, black and green banded chert, red jasper and quartz porphyry are well represented, pebbles consisting of tourmalinised rocks, and slaty or schistose types, are rare. The matrix consists predominantly of quartz, of sand size, with varying amounts of pyrite and other sulphides. Heavy minerals are represented by chromite, zircon and leucoxene.

The leucoxene is frequently uraniferous and investigations have shown that this is due to admixtures with brannerite. Brannerite is a minor but frequent carrier of uranium and is considered to account for much of the uranium in leach tailings.

Apart from gold and uranium, other valuable constituents occurring in minute traces are iridosmine and osmiridium. Phyllosilicates are present in the form of muscovite, pyrophyllite and chlorite. Carbon (thucholite) is notable in some of the conglomerates and occurs as granules of about 2 mm in diameter or less, and as seams that are composed of slender carbon columns arranged perpendicularly to the contacts of the seams. Such seams, which may be up to 15 mm thick, often mark the lower bedding planes of the so-called carbon-seam reefs in the West Rand Klerksdorp and Odendaalsrus areas.

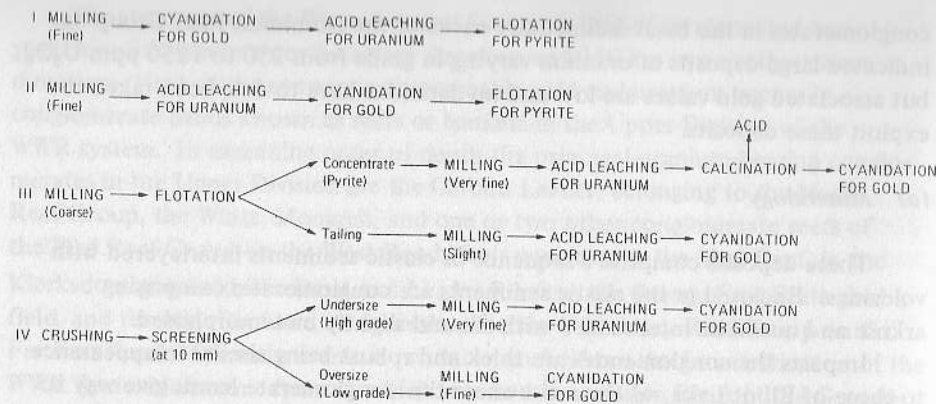


FIG. 26. Established procedures for the recovery of uranium, gold and pyrite from Witwatersrand ores.

Some forms of carbon contain abundant inclusions of uraninite, and often also of pyrite and gold. The enclosed uranium may not respond to conventional ore-processing techniques and this results in significant losses to plant residues.

(b) Processing flowsheets

The economy of processing these ores can be considered in context of three economic minerals, gold (which predominates), uranium and pyrite. A great deal of research has been done to examine various alternative routes for the whole ore processing to optimize the overall result. This work has resulted in developing a number of variations which have become established where the economics are appropriate. James (1975 [3]) illustrated the main overall established procedures and Fig. 26 is taken from his paper.

(c) Comminution

The original plants all used a feed material derived from the cyanidation process for gold recovery, either by recovering tailings from dumps or taking the gold plant residues directly. The only pretreatment consisted of a "neutral" filtration and washing, to remove as much of the residual alkaline, lime/cyanide reagents as possible before acid leaching. Except for one of the above illustrated routes, grinding is controlled for the gold cyanidation processes. The sizing of the ground ore is nominally 70% minus 74 μm and is generally finer than necessary for uranium only.

Since separation of pebble and matrix material cannot be achieved at an early stage of comminution, the grinding of both these conglomerate components to a maximum degree, consistent with economic operation, is usually accomplished using rod, pebble and ball mills. In view of the large tonnages involved and the economies of scale that are attainable, autogenous milling has proved particularly applicable in the milling of South African ores. The development of an effective centrifugal mill holds out some promise for the installation of mills underground. Mined ore would then be pumped to the surface for processing, thus obviating the present intermittent ore-hoisting cycle.

(d) Preconcentration or beneficiation

Screening of crushed products has proved effective in producing an enriched stream due to the more friable nature of some conglomerate matrix material. On one plant screening at 1 cm results in 67% gold and 60% of the uranium reporting in the fine fraction which constitutes 30% of the feed mass. The high-grade portion contains more of the thucholite, phyllosilicates and clays and benefits from additional grinding. The realization that pyrite is liberated at a relatively coarse grind has led to the introduction of flotation after a first stage of milling at some plants. Both the pyrite concentrate and the flotation tailings are further ground (to 90% minus 44 μm and 60% minus 74 μm , respectively). Both streams are then processed using a reverse leach procedure (acid leach for uranium followed by cyanidation for gold). This procedure considerably increased the effective capacity of the plants concerned. The association of uranium with carbonaceous material (thucholite) suggests the application of flotation to concentrate the uranium and this procedure has been very effectively applied by Anglo American Corporation on the East Rand (Ergo) and in the OFS (Joint Metallurgical Scheme-President Brand) to recover sulphur, gold and uranium from cyanide residue dumps containing, on average, as little as 0.53 g gold/t residue, 0.04 kg U/t, and 1.04% sulphur; 90% of the sulphur is recovered together with 70% of the uranium and between 70 and 90% of the residual gold. The relatively complex interrelated process flowsheets which have thus evolved are shown in Fig. 27 (from Ruhmen et al. 1977 [4]).

Wet high-intensity magnetic separation (WHIMS) is being tested on cyanide residues having low uranium grades to effect the recovery of some 50% of the gold in a concentrate comprising some 10% of the feed mass.

(e) Leaching

The ore is leached in pachucas for approximately 48 h in a slurry of 60 to 65% by weight of solids at 55 to 65°C. Acid is added by conductivity control to

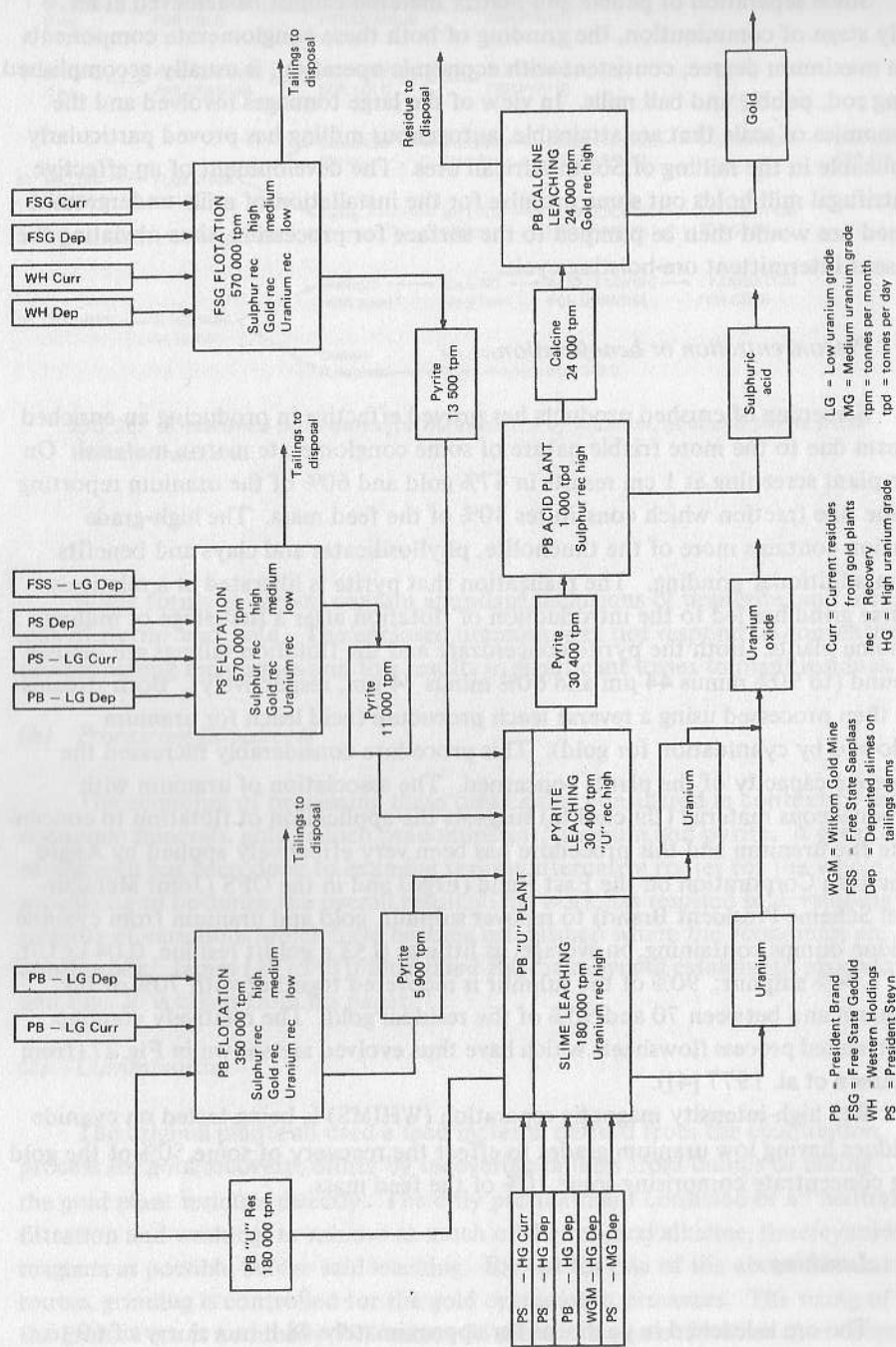


FIG.27. Orange Free State (OFS) joint metallurgical scheme.

maintain 5 g/litre free acidity (approximately 0.1 N compared with 1 N for Blind River brannerite). Crude pyrolusite is ground and added as the oxidant.

The acid-leaching process recovers 80 to 85% of the U_3O_8 . However, it should be remembered that this is from a feed containing only 0.025% U_3O_8 so that the final tail is only about 0.002%.

Basically all the uranium is recovered from the uraninite and the uranium in tailings reflects the presence of a small proportion of refractory minerals. Recovery has been increased in experimental work to the 90 to 95% range using more aggressive conditions such as pressure leaching and the "Felix", high ferric iron and acid leach.

With reference to the alternatives shown in Fig.18, in most of the earlier uranium plants acid leaching followed cyanidation (procedure I), mainly because this involved no change in the existing gold circuits. Prior acid leaching (procedure II) improves the subsequent extraction of gold. For some ores prior flotation (procedure III) can be used to produce a concentrate containing up to 80% of the gold, 80% of the pyrite and 25% of the uranium in less than 10% of the mass of the original ore. Because of the very small mass of this concentrate it can be milled very finely and treated intensively to yield very high recoveries of gold and uranium. The overall advantage is a saving in milling costs for the bulk of the ore. At other mines there is a size split of the crushed ore (procedure IV), which results in a fine fraction (approximately 30% of the original mass of ore) that contains two thirds of the gold and more than half the uranium. This fine fraction is given an intensive treatment again with overall savings.

(f) Solid-liquid separation

The separation of liquids and solids prior to concentration and purification has usually involved the use of stainless-steel rotary-drum filters followed by downflow sand-bed clarification. The application of countercurrent thickening aided by flocculating agents (Blyvooruitzicht), or the use of horizontal belt vacuum filters (Randfontein), are two recently introduced alternatives that are now generally preferred to rotary-drum filters in the design of new plants.

(g) Concentration/purification

Fixed-bed ion-exchange was initially employed to absorb uranium from solution with a later move being made towards the use of solvent extraction (Purlex process). Although solvent extraction required a more expensive clarification system for the leach solution, it brought about economies in reagent consumption and produced a purer product. In the pre-reverse-leach era, solvent extraction also avoided the frequent resin replacement found to be necessary in the fixed-bed ion-exchange process owing to cobalt-cyanide poisoning. In recent

years the development of the NIMCIX continuous up-flow fluidized-bed ion-exchange contractor has had a major impact on the choice of recovery flowsheets. The NIMCIX system has been applied on an industrial scale in combination with CCD at Blyvooruitzicht and has been adopted in combination with horizontal belt filers for the new plants at Stilfontein and Merriespruit. A Himsley continuous ion-exchange system will be included in the new Vaal Reefs South plant. To date only strong base resin circuits involving elution with 10% sulphuric acid and final purification of the eluate in a small solvent extraction plant have been used. (Eluex, Bufflex). A practice not followed anywhere else is the transportation of ADU slurry in road tankers from all the uranium plants to a central drying, calcining and packaging facility operated by the Nuclear Fuels Corporation of S.A. (Nufcor).

3. VEIN-TYPE DEPOSITS

Epigenetic low- and high-temperature hydrothermal deposition and redistribution of uranium into the cavities of host rocks has generally been considered to lead to the formation of so-called vein deposits. The source of the first uranium mined, these forms of deposits have continued to supply a modest proportion of yellowcake to the market. However, the discoveries of uranium in such deposits over the past ten to fifteen years in Australia and Canada have increased the known reserves to place them in the position of the second largest potential source.

Vein deposits are generally considered to be controlled in their formation by structural rather than chemical characteristics of the host rock. Characteristically the deposits contain significant coarse-grained to massive chunks of uraninite — usually described as pitchblende in this form — together with uraninite more finely dispersed within the rock matrix.

The pitchblende is typically low in thorium. In keeping with the hydrothermal origin some deposits have associated with the uranium a complex assemblage of metal sulphides containing copper, nickel and cobalt in economic amounts together with iron sulphides. However, other deposits can be almost free of metal sulphides (such as the Alligator river ores of Australia).

Uraninite is the dominant source of uranium in the unoxidized zone. However, trace to minor amounts of complex uraniferous titaniferous oxide material have been shown to be present in Canadian, Australian and European ores. Various “secondary” hexavalent uranium minerals are found above the water table.

Although the mineralization is not confined to a particular rock type, in keeping with the expected structural environment it is not uncommon to find it associated with brecciated material and argillaceous, schistose metamorphic rock with quartz and mica as major matrix minerals or alteration products, particularly chloritic. Haematite is common and calcite can be significant but is not always present.

3.1. Canada — northern Saskatchewan

A recent review (McMillan 1977) summarizes this deposit, and this report has adopted this subdivision of “classical” vein deposits and “unconformity” vein deposits in Canada.

3.1.1. Classical vein deposits

The Port Radium deposit was the first worked for significant production of uranium as the main product, and continues to have a small production of uranium and silver from the Contact Lake Mine. However, most present-day production is from the Beaverlodge area: the Ace-Fay-Verna deposits; the Bolger, Nicholson and Martin Lake deposits, and the Gunnar deposit.

The Ace-Fay-Verna deposits occur mainly in the footwall of the southeast dipping St. Louis fault. Breccia ore occurs in zones ranging from 0.3–1.5 m thick while vein-type ore is found within 90 m of the fault. These deposits have a strike length of more than 4500 m and are known to a depth of 1650 m. The Gunnar deposit is pipe shaped and plunges about 45° south for 670 m having a maximum diameter of 135 m. The deposit is located close to Zoemel and St. Mary's channel fault.

The Port Radium mine, which produced silver and radium in the 1930s, is of historical interest. The major constituent of the ore was uranium but had no use at that time and was discarded. In 1942 Eldorado re-opened the mine for the recovery of uranium. Eldorado Nuclear have been producing uranium in the Beaverlodge area since the early 1950s. The Beaverlodge mill was expanded in 1957 to 2000 t/d of ore and has undergone several modifications to its flowsheet. Currently, the treatment rate is 900 t/d with plans to increase production to 1630 t/d by 1979. Eldorado are exploring several prospects near to its Beaverlodge property and at least two will be further developed. Nearby, in joint venture with PNC-Japan, Eldorado are assessing some 500 anomalies found by airborne radio-metric surveys carried out in 1973.

(a) Mineralogy

Uranium occurs mainly as pitchblende. The Gunnar deposit also contains minor uranophane, particularly in the deeper levels of the mine.

Most of the ores have relatively simple mineralogical characteristics, the pitchblende being associated with calcite, quartz and minor pyrite. The Bolger, Nicholson and Martin Lake ores, however, are complex and commonly also contain copper sulphides, cobalt-nickel sulpharsenides, silver, gold and selenides. The Gunnar ore contains quartz, chlorite, kaolinite, chalcopyrite, pyrite and galena. Carbonate content ranges from 5–8% in the feed to the Eldorado-Beaverlodge plant.

(b) Comminution

The ore is extremely hard with a Bond work index of about 20. The run of mine ore is crushed underground to minus 15 cm. The coarse ore is ground in a double-compartment semi-autogenous mill which operates in closed circuit with a cyclone and classifier in series. The latter, in turn, operates in closed circuit with two ball mills. The classifier overflow is about 25% solids and 75% minus 200 mesh. Plant solution, containing 100 g/litre of dissolved carbonate and sulphate salts, is used in the milling circuit.

(c) Preconcentration or beneficiation

About 9 t/d pyrite concentrate is produced by flotation. The ore contains about 0.3% sulphur, which would consume over 3 kg sodium carbonate/kg sulphur. Four banks of eight flotation cells are used for roughing and for cleaning. Four cells are used for cleaning.

(d) Leaching

The sulphide concentrate is thickened, filtered to remove carbonate solution and acid leached for uranium in air-agitated pachucas using sodium chlorate as oxidant. The leached slurry is filtered twice and the uranium precipitated as a crude magnesium diuranate which is added to the carbonate leaching circuit. Flotation tailings are thickened to 55% solids and leached for uranium with sodium carbonate-bicarbonate solutions in pachucas that have been fitted with mechanical agitators. Retention time is four days (four banks of six pachucas) for a uranium dissolution of 92%. Oxidant is added in the form of pure O_2 from an oxygen plant. Leach temperature is 90°C.

(e) Solid-liquid separation

After cooling the leached pulp to 65°C in pulp-to-pulp heat exchangers, pregnant solution is recovered in a two-stage filtration circuit utilizing rotary drum vacuum (string) filters.

(f) Concentration/purification

The filtered leach liquor is concentrated in a two-stage evaporation circuit, termed steam stripping. About 50% of the sodium bicarbonate is destroyed in the steam stripper, the balance is removed by calcium hydroxide addition. A thick $CaCO_3$ slurry is recovered in a Lamella thickener and returned to the

carbonate leach circuit. Sodium diuranate is precipitated directly from the clarified pregnant solution.

3.1.2. "Unconformity" vein deposits

These are those deposits worthy of separate summary description as given in the following paragraphs.

Cluff Lake

Drilling has identified three orebodies at Cluff Lake within 5 km of each other on the south end of the Carswell circular structure. The D orebody consists of a flatish cigar-shaped zone 140 m long by 25 m wide. Average grade is 5–10% U_3O_8 with portions assaying up to 65% U_3O_8 . The N orebody is about 1200 m long, 120–200 m wide and 50–150 m deep and is located in a zone of alternating bands of gneiss. The Claude orebody is 600 m long, 200 m wide and 90 m deep and is located entirely within basement rocks. The grade of the N and Claude orebodies is about 0.5–1.0% U_3O_8 .

Amok-Cluff Lake plans to start mining the D orebody in late 1979. The mill will process only 80 t/d of ore but produce 1500 t U/a. After 1982 the mill will be expanded to treat much larger tonnages of ore from the Claude and N orebodies, to maintain the same uranium production rate.

(a) Mineralogy

Uranium minerals in the D orebody are uraninite and pitchblende and coffinite occurs along shear zones, with the highest concentration near the intersection of faults. In the Claude orebody uraninite and coffinite are found in flat-lying shears in quartzite gneiss.

Gold is an accessory mineral in the D orebody (0.5 to 1.0 oz/t) which also contains small amounts of selenium, cobalt, bismuth, lead, and various selenides and tellurides. All three orebodies contain varying amounts of pyrite, galena, pyrrhotite and chalcopyrite. Garnet, marcasite, cordierite and silliminite is found in the N and Claude orebodies.

(b) Comminution

The degree of comminution required is 80% minus 48 mesh (50% minus 65 mesh). This will be achieved by crushing and single-stage rod milling in closed circuit with classifiers.

(c) Preconcentration or beneficiation

Gravity concentration is possible on the D orebody to produce a concentrate with a uranium concentration of 45% U_3O_8 .

(d) Leaching

Acid leaching will be used in mechanical agitators at 45–55°C with sodium chlorate addition. Pulp density will be 60–70% solids decreasing sharply as uranium dissolution occurs.

(e) Solid-liquid separation

Two stages of horizontal belt filters will be used for solid-liquid separation. No classification is envisaged.

(f) Concentration/purification

Uranium will be precipitated directly from the acid leach liquors as magnesium diuranate, without prior concentration or purification by ion-exchange or solvent extraction.

Rabbit Lake

The Gulf deposit is located in the Wollaston Fold Belt on the up-thrown side of the low-angle Rabbit Lake fault, which has thrust basement rocks over the Athabasca Formation. The orebody occurs as fracture and breccia fillings of massive pitchblende and is wholly within a zone of chlorite-quartz-dolomite alteration. The relatively small, high-grade core is surrounded by a mantle of lower grade material. Average grade is 0.30% to 0.40% U_3O_8 .

The Gulf-Uranerz open-pit operation at Rabbit Lake achieved full capacity (1800 t/d) late in 1976. The mine and mill are operated on a seven-day-week, two 12-hour shift basis. Workers are flown out on alternate weeks to their permanent homes in several northern Saskatchewan communities as far south as Saskatoon. Of the total work-force of 260 employees, some 100 are on site at any one time. Exploration has continued in the Rabbit Lake area and reports are that four additional deposits have been discovered recently.

(a) Mineralogy

Uranium occurs as pitchblende, carnotite and uranophane. The main gangue minerals are haematite, calcite, dolomite, vuggy quartz, chlorite and clay minerals. Minor amounts of pyrite, chalcopyrite and galena occur in small quantities.

(b) Comminution

The ore is ground in two stages in an autogenous mill followed by a ball mill in closed circuit with cyclones.

(c) Preconcentration or beneficiation

Preconcentration is not used but trucks from the pit are monitored radio-metrically and the ore, which tends to be erratic in grade, is blended prior to milling.

(d) Leaching

The cyclone overflow passes directly to the acid leach circuit which operates at a pulp density of 55% solids. Thus, there is no preleach dewatering stage as practised in the Elliot Lake area.

(e) Solid-liquid separation

The leached pulp reports to a six-stage CCD conventional thickener circuit. The underflow density achieved is 50% solids. The overflow contains 75 ppm solids and passes first to a thickener for primary clarification. This thickener is exposed to the weather and has a submerged overflow weir to ensure that ice and floating slime are not passed onto the second clarification stage which comprises downflow clarifiers.

(f) Concentration/purification

Uranium recovery is by solvent extraction, an ammonium sulphate strip and ammonia precipitation.

Key Lake

The two orebodies discovered by Uranerz in the Wollaston fold belt are located within a relatively long northeast-trending fault zone dipping steeply

northwest. The Gaertner orebody is 1500 m long with a vertical thickness between a few metres and 80 m and a width of 10–40 m. The grade ranges up to 45% U_3O_8 and 45% Ni. The Deilmann orebody is 800 m long, 10–100 m wide and up to 70 m in vertical thickness. Drilling is still continuing on both deposits.

The recent discovery by Uranerz of two major deposits at Kay Lake has stimulated a rush of exploration activity along the eastern limit of the Athabasca outcrop area and several more discoveries have resulted. In February 1978 NUMAC-Esso announced a major discovery at Midwest Lake which is reported to extend 850 m parallel to a substructure in the Basement rock. This discovery has given exploration geologists a new and deeper target area.

(a) Mineralogy

Mineralization occurs in fault and fracture zones affecting the upper Athabasca sandstone and the metasediments of the Wollaston fold belt and the cover is of glacial origin. The lithology of the country rocks consists of quartzites, chlorite sericite schist, graphite schist, biotite gneiss and pegmatitic material beneath glacial sand and gravel.

However, the mineralization is massive and rich with almost no gangue mineral in the ore zone. Grades of up to 45% U_3O_8 and 45% nickel and many metres thick are reported for the Gaertner orebody and up to 20% U_3O_8 and 25% nickel for the Deckmann orebody.

Uranium minerals are pitchblende and coffinite. Two distinct types of pitchblende have been identified. Type I is characterized by its radiating texture and high reflectivity. Type II is synonymous with sooty pitchblende and is often characterized by shrinkage cracks and is generally intensively inter-grown with the nickel phases.

Nickel occurs as gersdorffite, millerite and niccolite, with bravoite as an accessory. Pyrite, sphalerite, chalcopyrite and galena occur in minor amounts.

(b) Treatment aspects

No information is yet available on any proposed treatment techniques.

3.2. Australia — Alligator Rivers Uranium Province

There are four different proprietary interests who have announced discoveries of substantial reserves for future mining and treatment. Their deposits are distributed along an approximately rectangular area, running northeast to southwest for a length of approximately 100 km and a width of 30 km. One deposit (Jabiluka) lies approximately in the centre of the NE-SW line, about 2° longitude and 135 km due east of Darwin.

The total proven reserves of the province at present exceed 330 000 t U_3O_8 , in ores whose grades are all classified as high grade, so far as economic grades are considered (greater than 0.15%), and some additional reserves are present in lower grade material (less than 0.1%).

Economic mineralization is confined to the formation in the lower Proterozoic now known as the Cahill formation, considered equivalent to the Koolpin and Golden Dyke formations. The mineralogy of the host rocks is that associated with the following typical sequence, and their surface weathering products. Muscovite-chlorite schists; carbonaceous (graphitic) schists; quartz-chlorite schists; breccias and minor dolomites and cherts. Primary uranium mineralization is essentially uraninite. However, minor amounts of refractory minerals occur to a more or less degree and weathering has resulted in the number of secondary minerals in the near surface material. The following are more specific descriptions of the deposits.

3.2.1. Nabarlek

This is the most northern deposit and part of the lease of Queensland Mines Ltd. The initial design capacity of the mill will be 190 t/d of ore (900 t U/a).

(a) Mineralogy

Sections of almost pure pitchblende occur (72% U) in irregular lenses with intersections to one metre thickness. Smaller pods of massive mineral and high-grade material occur in a more extensive body of lower-grade mineralization. The massive high grade is intimately associated with the siliceous gangue minerals in which mica and chlorite predominate. The pitchblende is characteristically interleaved with mica flakes, forming a continuous body with the segregated mineral.

In lower-grade primary mineralization, pitchblende is fine grained, and there are indications of colloform supergene material. Chlorite is a dominant gangue mineral with some haematite and sericite. Sulphides are minor minerals (less than 0.5%). No gold occurs in this ore.

In the oxidized or weathered zone, which extends from 5 to 15 m in depth, secondary uranium minerals predominate and kaolinite occurs with quartz, muscovite and chlorite. The uraniferous minerals consist of curite, sklodowskite, rutherfordite and traces of kasolite, with uraninite appearing as the unoxidized zone is approached.

In both primary and secondary ore, there are small amounts of unnamed refractory uranium — titanium minerals and uranium-silicate minerals and some uraniferous zircon. The average grade of this orebody, with a cutoff at 0.1%, is approximately 2.4% U_3O_8 containing approximately 9500 t U_3O_8 and with

cutoff at 2.0% the ore would still contain approximately 8250 t U_3O_8 at an average grade of 9.3%.

(b) Comminution

Comminution will be by crushing to minus 13 mm primary rod milling and secondary ball milling to produce a feed to leaching at a grind of minus 50% minus 200 mesh.

(c) Preconcentration or beneficiation

The nature of the Nabarlek ore offers the option of using gravity concentration to recover a significant proportion of pitchblende in a low mass fraction exceeding 50% U_3O_8 in grade. Although the tailings so produced will be relatively high grade such an approach is attractive in that it could generate an early cash-flow. This option however is not proposed to be used.

3.2.2. Ranger (Jabiru)

Ranger Uranium Mines have proposed mining two orebodies of approximately equal size. One is located near the surface and will be mined by open-pit methods. The other is somewhat deeper and part will be mined by underground methods. The initial design capacity of the mill will be 4000 t/d ore (2500 t U/a).

The cutoff figure, suggested for mining to ore stockpiles, is 0.05% U_3O_8 and at this figure the minable reserves are estimated to be 20 000 000 t ore with average grade of 0.25% U_3O_8 from the number one orebody and 23 000 000 t at 0.22% from number three orebody. Lower-grade material will be mined to stockpile.

(a) Mineralogy

Some high-grade intersections have been encountered, but in general the mineralogy is more similar to the lower-grade primary ore at Nabarlek. However, the gangue from the host rock contains less muscovite and more predominance of chlorite and quartz. The weathered-lateritic material contains much kaolinite but it is a relatively small proportion of the total ore. Economic amounts of gold have not been reported.

(b) Comminution

The ore will be crushed to -20 mm and ground in a rod mill and closed-circuit ball mill to a nominal 50% minus 74 μm . Preconcentration has not been considered applicable.

3.2.3. Jabiluka

Pancontinental discovered Jabiluka One in late 1971 while Jabiluka Two was identified in 1973. The two orebodies together represent the largest known uranium deposit in the world. There has been speculation that the two orebodies will eventually be connected as additional holes are drilled. Mining will be by open-pit methods.

The large orebody has geological reserves of 204 000 t U_3O_8 . The mining plan proposed for the two orebodies is estimated to produce 36 000 000 t ore with an average grade of 0.44% U_3O_8 using a cutoff of 0.05%. There is an estimated 530 000 t ore containing 50 g/t gold. Initial design capacity of the mill will be 3200 t/d of ore (2500 t U/a).

(a) Mineralogy

The mineralogy is described in the recently released environmental study as follows. Uraninite and pitchblende essentially constitute all of the uranium mineralization at Jabiluka.

Mineralization is contained in the following series of host rocks:

The upper graphite series which are described as the graphite schists, chlorite graphite breccia and dolomite.

The main mine series, described as graphite schists, quartz-muscovite chlorite schists; chlorite schists (plus or minus graphite); cherts; dolomites; schist breccias.

The lower mine series 1, described as quartz, muscovite chlorite schists.

The lower mine series 2, described as chlorite schist (plus or minus graphite); chlorite feldspar schist; schist breccias and dolomite.

This host rock series is more or less typical throughout this province. Some of the series are reported to contain minor amounts of sulphide, the graphite schist in particular containing up to 5% pyrite.

However, the average grade of sulphide is much lower and more similar to that reported for Nabarlek. Higher-grade intersections have also been encountered in this orebody, and gold has been detected in relatively high-grade concentrations in part of the orebodies.

(b) Comminution

A conventional circuit. The crushed ore will be -25 mm and ground ore 50% minus 74 μm . Preconcentration is not being considered.

3.2.4. Prosposed treatment routes

For the three orebodies — Jabiru, Jabiluka, Nabarlek — the following summarizes the treatments proposed and their features:

(a) Leaching

	Jabiru	Jabiluka	Nabarlek
Pulp density (% solids)	55	56	55
Acid consumption (kg/t)	50	45–50	90
Pyrolusite oxidant (kg/t)	5–6	5–6	10
Temperature ($^{\circ}\text{C}$)		35	40–50
Time (h)	24	36	36
pH		1.8	1.5

Although the leaching of these ores is relatively straightforward, considerable care is required to decide the best combination of pH, temperature and time of leaching, in order to optimize the consumption of acid against recovery of the uranium. The reactivity of the mica and chlorites in particular is quite sensitive to pH, within the range used for dissolution of uraninite. The Nabarlek secondary ore, containing mainly secondary minerals, cannot satisfactorily be leached at pH 2.0. In that ore the highly reactive silicates have already been weathered and leaching at pH 1.0, although not the optimum, would still only have resulted in an acid consumption of 75 to 78 kg/t. However, for the primary Nabarlek pH 1.0 proved most expensive, so far as acid consumption was concerned, reaching 200 kg/t after 15 h compared with 80–90 kg/t at pH 1.5 between 20 and 24 h. Satisfactory extraction could be obtained from the primary ore from pH 1.5. This was chosen as the most satisfactory compromise for treating the mixture of both ores.

Presumably the higher acid consumption required for the primary Nabarlek ore, compared with the acid consumption proposed for the Jabiru and Jabiluka ores, is due to a combination of the requirement for additional acid and oxidant in order to attack the much higher-grade massive uraninite, and the sensitivity of the silicates in the primary ore to the acid conditions. It was calculated that a temperature of 45 $^{\circ}\text{C}$, proposed for treating Nabarlek, would be reached naturally, by the energy used in comminution, combined with the heat of dilution of the concentrated acid and of course the tropical environmental temperature. It is interesting to note, therefore, that the proposal for Jabiluka indicates that steam injection will be used to maintain a temperature of 35 $^{\circ}\text{C}$.

An interesting feature of the leaching of the Jabiru and Nabarlek ores is the relatively high tailings which test work indicates will result. Thus, leaching recovery from Jabiru is not expected to better 90%, and some work suggests it may be less, with tailings of 0.03 to 0.04% U_3O_8 . From Nabarlek the leaching will result in recoveries of 95–98% because of the high grade nature of the ore, but tailings of around 0.05 to 0.1% U_3O_8 are expected. The above feature of these ores compares with a similar feature observed from the South Alligator River ores which were treated in the 1950s and the 1960s. Work on this problem has revealed that there is a proportion of uranium distributed ubiquitously in the light silicate minerals and a proportion of heavy refractory minerals, where uranium is combined with titanium and phosphate and a uranium silicate mineral. Some uraniferous zircon was also detected.

It is interesting to note that the final tailings from Jabiru could contain approximately 20 000 t U_3O_8 , a resource of some magnitude.

(b) Solid-liquid separation

The rheological properties are such that conventional leach pachucas may be used for leaching and continuous counter-current washing in thickeners (CCD) for solid-liquid separation.

(c) Concentration/purification

The high grade of the Alligator Rivers ores will result in pregnant solutions with uranium concentrations as much as 10 times (at Nabarlek) higher than is usual in uranium ore processing practice. Solvent extraction using tertiary amine solvent is favoured for uranium recovery and purification. Pre-acidification of the amine is required under such conditions, because of the high uranium tenor of the pregnant solution.

4. GRANITIC DEPOSITS

Uranium mineral dispersed in low-grade concentrations through granitic rock rather analogous to the so-called porphyry copper deposits. Crocker Well in Australia in which brannerite is dispersed in granite has been the subject of study but the major deposit of interest is that at Rössing.

4.1. Namibia (South West Africa): Rössing

The Rössing deposit is situated along the northern limb of a complex synclinorium in the central portion of the Damara orogenic belt. The bulk of the uranium mineralization is obtained in alaskite that is preferentially emplaced in the heavily folded gneisses, schists, marbles and limestones of the Khan and Rössing Formations.

The Rössing deposit in South West Africa is the world's first low-grade, disseminated uranium occurrence in granitic rocks to reach commercial development. The investigation of the deposit by Rio Tinto, during the period 1966–1973, proved the existence of a large low-grade deposit amenable to low-cost open-pit mining methods and relatively simple metallurgical processing techniques. Construction of the mine and mill facilities began in late 1974 and the plant was commissioned during 1976/77.

The success of the Rössing venture has stimulated exploration groups worldwide to look for similar massive, granitic, uranium deposits. In northeastern Brazil disseminated uranium mineralization has been found in granites and biotite schists in the Serido geosyncline. In the USA, exploration efforts are under way in the potential granitic districts of Montana, Idaho, Wyoming and Colorado. While a major discovery has not been made, evaluation results are encouraging.

(a) Mineralogy

Uraninite is the dominant primary uranium mineral in the Rössing deposit. It is included in quartz, feldspar and biotite, but also occurs interstitially to these minerals or along cracks within them. A minor proportion of the uranium is contained in betafite. The primary uranium minerals, uraninite and betafite, gave rise to secondary minerals that are usually bright yellow. These occur in situ, replacing the original uraninite grains, or commonly along cracks as thin films or occasional discrete crystals. Beta-uranophane is the most abundant of the secondary uranium minerals. (Primary uranium minerals in the Brazil granites are uraninite and urano-thorianite. Among the secondary minerals the most abundant is beta-uranophane. Meta-autonite, metatorbernite and francolite are also present).

Alaskite is a granitic rock composed almost entirely of quartz and potassium feldspar. Zircon, calcite and apatite are relatively widespread while monazite, pyrite, chalcopyrite, bornite, molybdenite, arsenopyrite and fluorite are often observed. The oxides magnetite, haematite, and ilmenite are occasionally encountered. Betafite has a high niobium and titanium content and contains small amounts of tantalum and tungsten.

(b) Comminution

In spite of the generally small grain size of the uraninite (100–300 mesh) it is only necessary to grind to minus 6 mesh. The uraninite is accessible to leaching at a much coarser grind than the expected liberation size because in many instances it occurs within cracks, in the quartz and feldspar, along which leach solutions can penetrate. Conventional crushing and open-circuit rod mill grinding is practised at Rössing.

(c) Preconcentration or beneficiation

Because of the variable nature of the ore, investigations are under way at Rössing to introduce the use of scintillometers for monitoring trucks as a means of rejecting waste that is unavoidably introduced during mining.

(d) Leaching

Both primary and secondary uranium minerals are leached relatively easily at a surprisingly coarse grind for the reason mentioned above. Leaching is also facilitated by the fact that the alaskite is somewhat porous and fairly extensively cracked and that uranium occurs between flakes of biotite and chlorite that are easily exposed during grinding. Betafite is a highly refractory mineral and not amenable to conventional acid leaching. Acid leaching in mechanical agitators is employed at Rössing. Pulp density is > 75% solids and temperature 45–50°C. Manganese dioxide is added as oxidant and the pH is closely controlled using electrodeless conductivity probes. Careful attention is paid to the calcite content of the feed to the plant to ensure optimum acid consumption levels in the leach.

(e) Solid/liquid separation

A sand/slime separation is first carried out in hydrocyclones. Sands are washed in a two-stage countercurrent circuit using Rotoscopes. Slimes are washed in a CCD circuit based on conventional thickeners. The original CCD circuit of three stages is being converted to a five-stage circuit to improve the overall effi-

ciency of uranium recovery. Weathering of the feldspar has given rise to the presence of clay minerals such as montmorillonite. During crushing, the clay-mineral fraction almost immediately becomes slime. Normally the proportion of clayey material in the feed is such that it helps to keep the coarser material in suspension in the leach and does not interfere in the CCD circuit. When present in abnormally high concentrations it has a detrimental effect on these operations and its content in the feed therefore has to be carefully controlled.

(f) Concentration/purification

Uranium is recovered at Rössing from the unclarified pregnant solution by an Eluex/Bufflex circuit comprising continuous ion-exchange (Porter system) and solvent extraction (conventional mixer-settlers). Ammonium diuranate is calcinated in a multiple hearth roaster.

4.2. Australia — Crocker Well

Situated close to the old Radium Hill mine in South Australia this deposit has been subject to study intermittently since the mid 1950s. However, there are difficulties establishing adequate reserves and mining has never proceeded.

It is reviewed here because of the notable success with flotation demonstrated in experimental work on laboratory and pilot scale and because the orebody is again the subject of exploration study.

(a) Mineralogy

The uranium occurs as brannerite, described in early mineralogical reports as the absite form of brannerite. It occurs in brecciated granatoid rocks, the associated gangue being quartz, biotite, apatite and rutile. Although selective sampling produced samples containing 0.1% U_3O_8 the average grade of the ore is low, approximately 0.04% and some form of beneficiation is recognized as being required in order to leach the refractory brannerite.

(b) Treatment characteristics

Flotation was very effective in concentrating the brannerite from the Crocker Well ore. Like cassiterite, brannerite is brittle and tends to slime when grinding, but release of the mineral in the case was sufficiently complete at 350 μm to treat the ground ore and achieve good recoveries.

Flotation was effected with mixtures of sulphonated whale oil, linseed fatty acids, diesel fuel oil and cresylic acid, reagents similar to those used to treat Radium

Hill ore. From ore containing approximately 0.05% U_3O_8 , concentrates were produced containing approximately 2.0% U_3O_8 with recovery of approximately 84%. This represented a very considerable rejection of material prior to the application of a leaching process. Unfortunately almost all the apatite was reported in the uranium concentrate and was the major mineral together with significant amounts of biotite. Selective removal of biotite, as a first stage of flotation, is a method suggested for controlling this mineral, but no method could be found for separating or depressing the apatite. However, it is believed that satisfactory leaching procedures could be developed for this high-grade concentrate.

5. PEGMATITES, CARBONATITES AND SYENITES

One significant economic deposit of pegmatite type occurs in Canada and a minor quantity of uranium has been located in carbonatites in Africa and India and in syenite in Greenland.

5.1. Canada — Pegmatites Bancroft

Uranium deposits in the Bancroft area, southeastern Ontario, Canada, occur in pegmatitic granite dykes in a region of metamorphic and metasomatic rocks. By comparison with many other pegmatitic occurrences some of the Bancroft orebodies are unusually large, persistent in depth and relatively rich in uranium minerals.

Several mines and mills were in production in the Bancroft area at one time but production ceased in 1974. Production has again started with the recent reopening of the Faraday mine.

Madawaska Mines completed rehabilitation of the Faraday mine and mill in August 1976. Full capacity of 1360 t/d of ore has now been reached. In addition to the Faraday deposit a number of smaller deposits, some amenable to open-pit mining, could possibly be exploited by the establishment of a central custom treatment plant.

The Bancroft area is currently attracting the interest of several companies exploring for uranium including Imperial Oil, Inco, Kerr Addison and Madawaska Mines.

(a) Mineralogy

Uranium is present as uraninite and uranothorite as disseminated, discrete crystals within the pegmatite. Minor betafite, allanite, zircon, and fergusonite occur. Signs of the presence of ore mineralization include haematite staining, an

abundance of ferromagnesian minerals and the presence of magnetite. Uranium grade is in the range 0.05–0.10% U_3O_8 .

(b) Comminution

Conventional crushing and two-stage grinding in closed circuit with rake classifiers is practised by Madawaska. Rod and pebble mills produce a leach feed with a grind of 80% minus 100 mesh. No sorting or preconcentration is practised.

(c) Leaching

Acid leaching is conducted in air-agitated pachucas at ambient temperature and a pulp density of 65% solids. Sodium chlorate is added as oxidant. Dewatering in thickeners prior to leach. 38 hours residence time.

(d) Solid-liquid separation

Two-stage rotary drum vacuum filters (string discharge type). Clarification is by rotary drum precoat filter and pressure sand clarifiers.

(e) Concentration/purification

Fixed-bed ion-exchange using strong-base resin and sodium chloride elution. Uranium is precipitated as magnesium diuranate directly from the eluate.

5.2. South Africa — Carbonatites — Phalaborwa

The Phalaborwa Complex in the northern Transvaal, South Africa, consists of pyroxenite, syenite and a centrally located core of transgressive carbonatite which is surrounded by a serpentine (olivine-magnetite-apatite) rock to which the name phoscorite has been given. Small concentrations of uranium and thorium occur in the phoscorite and carbonatite.

Copper, apatite and vermiculite are mined on a large scale by the Palabora Mining Company (PMC) and by Foskor using open-pit methods.

Uranium has been recovered by PMC since 1971 as a byproduct from a heavy-minerals gravity concentrate produced from the copper-concentrator tailings.

(a) Mineralogy

Copper occurs principally as chalcopyrite. Bornite, chalcocite, cubanite and valleriite also occur. Uranium and thorium are present in uranothorianite. Baddeleyite (zirconium oxide) is also present in small amounts.

(b) Comminution

Comminution of the run-of-mine ore to about 50% minus 200 mesh is adequate to liberate the copper sulphides for flotation. The flotation tailings, from which the uranothorianite is ultimately recovered, are not ground any finer.

(c) Preconcentration or beneficiation

The tailings from the copper concentration, after desliming in hydro-separators removal of magnetite and further desliming in hydro-cyclones, are subjected to gravity concentration in Reichert cones to give a 40–50 fold concentration of heavy-minerals fraction. This gravity concentrate is fed to shaking tables, where separation of a uranothorianite concentrate from the other heavy minerals is achieved.

(d) Leaching

The uranothorianite concentrate is leached with hot nitric acid resulting in dissolution of both the uranium and the thorium but not the zirconium oxide. Fumes (oxides of nitrogen) from the leach are scrubbed with ammonium hydroxide.

(e) Solid-liquid separation

The hot leach pulp is filtered on tilting-pan vacuum filters. The unclarified pregnant liquor is settled and then clarified in a centrifuge before being fed to the solvent extraction plant.

(f) Concentration/purification

Uranium and some thorium are extracted with 10% TBP in a six-stage mixer-settler bank. The loaded solvent is fed to a four-stage unit where traces of thorium are scrubbed out with pure uranyl nitrate solution. Uranium is then stripped with warm water in an eight-stage unit and precipitated as ADU with ammonia.

5.3. Greenland — Syenites — Ilimaussaq

Uranium and thorium mineralization is found at Kvanefjeld in the northern part of the Ilimaussaq intrusion near Narsaaq in South Greenland. The Ilimaussaq intrusion is composed of an old series of syenites and granites and a younger layered series of nepheline syenites. Uranium and thorium mineralizations are found in the latter, close to and within contact zones between sheets of lujavrite and altered volcanic cover rocks.

The Pilanesberg intrusive complex near Rustenburg, Transvaal, South Africa, was emplaced at the contact between the mafic and acid phases of the Bushveld Igneous Complex. Uranium, thorium and rare-earths occur in tuffs while uranium, niobium and other minerals are associated with foyaïtes.

Studies on the extraction of uranium from the ore at Kvanefjeld have been made over a number of years and a small pilot plant is being operated at the Risø Research Establishment in Denmark.

Laboratory studies have been undertaken on the Pilanesberg tuffs and foyaïtes but results are not encouraging.

(a) Mineralogy

An important part of the uranium and thorium in the Kvanefjeld lujavrite is contained in steenstrupine, monazite and pigmentary material. Uranium grade is of the order of 300 ppm U_3O_8 . The niobium, uranium and barium that are present in the Pilanesberg foyaïtes occur in pyrochlore and allanite. Pyrochlore fills veinlets and intergranular spaces, where it is mixed in a highly complex manner with zircon, pyrophanite and titanite.

(b) Treatment

There is no specific proposal on the treatment. However, the uranium in the Kvanefjeld ore is trapped in refractory minerals and the only approach that has appeared promising to date is the use of a sulphating roasting technique.

6. URANIFEROUS SHALES

There are a number of extensive but low to very low grade deposits of shales. Three such deposits are summarized here.

6.1. Egypt — Quatrani

Uraniferous shales characterized by the presence of clays occur in the Lower Sandy, Middle Clayey and Upper Formation of the G. Quatrani Formation in the northern part of the Western Desert of Egypt.

Developments are at the early stage of exploration. Some metallurgical test work has been published.

(a) Mineralogy

Uranium is associated with humic organic matter and is very low grade (0.03% U_3O_8).

Other minerals are dolomite, calcite, gypsum, haematite, goethite, quartz sand, native sulphur, sericite, montmorillonite and soluble salts.

(b) Comminution

Grinding to minus 10 mesh is usually adequate.

(c) Preconcentration or beneficiation

Laboratory-scale magnetic separation yielded slight concentration of U and Fe_2O_3 . Tabling proved ineffective and separation of uranium, gypsum, carbonate and iron oxide components is unlikely by physical means.

(d) Leaching and uranium recovery

Since uranium is present in the hexavalent state, both acid and alkali lixiviants are effective. However, stoichiometric additions of Na_2CO_3 are required to react with the gypsum. The presence of clays can be expected to complicate pulp agitation, decantation and thickening.

6.2. Sweden — Ranstad

Large, low-grade uranium deposits occur in bituminous shales which comprise part of a series of sediments belonging to the upper Cambrian ore.

A small uranium mill was put into operation at Ranstad in 1965. Shortly afterwards the plant was shut down but the facilities have since been used for pilot-plant development work. In 1975 LKAB took over the facilities and have undertaken a detailed feasibility study for a mine and mill to recover 1500 t U/a.

Although the economics appeared favourable at current world prices of uranium the project was vetoed by local authorities. Latest plans for Ranstad include a link-up between Boliden and LKAB to recover as many minerals as possible. Possible co-products include Al, K, Mg, P, S, U, V, Mo and Ni. The shale also has fuel potential. Mining companies are now intensifying their uranium exploration efforts for uraniferous shales in northern Sweden.

(a) Mineralogy

Uranium is associated with organic material. Grade is between 0.025 and 0.033% U_3O_8 . Shale is fine grained and consists mainly of quartz, illite and feldspars. Some 22% comprises organic matter and another 13% consists of pyrite.

(b) Comminution

Primary crushing is underground. Coarse float product from heavy media separation is screened out and comminuted in a hammer-mill.

(c) Preconcentration or beneficiation

Heavy media separation is used to reject limestone.

(d) Leaching and uranium recovery

Minus 2 mm crushed product is mixed with H_2SO_4 in curing drums prior to stockpiling. Oxidation and bacteria initiate leaching which is continued in vats to give a leach period of six days. Acid consumption is 60 kg/t shale.

The coarse product obviates settling and filtration problems.

Leach liquor is filtered and uranium extraction accomplished using a tertiary amine followed by sodium carbonate, or ammonium sulphate stripping. Yellow-cake precipitation using sodium hydroxide.

6.3. USA — Chattanooga

Chattanooga shale formations are generally around 10 m thick and consist of the Gassaway (upper) and Dowellton (lower) members. Their total uranium content has been estimated to be more than 10×10^6 t U_3O_8 at an average concentration of about 40 ppm U_3O_8 .

A paper exercise was done in the 1950s on the establishment of a 20 000 t/d operation but the economics was not favourable at that time. In view of the higher uranium prices now prevailing, Mountain States Mineral Enterprises of Tucson, Arizona, have recently been contracted by the Grand Junction office of the US Department of Energy to undertake work that will help determine the technical, economic and environmental feasibility of large-scale uranium production from Chattanooga shale.

(a) Mineralogy

Some 80% of the uranium in the Top Black shale is associated with organic matter. Silicate, aluminium, iron and sulphur minerals are present in substantial amounts.

(b) Comminution

Primary jaw crushers, secondary gyratory and tertiary hydrocone crushers followed by dry-rod milling were proposed.

(c) Pretreatment

Roasting generally enhances subsequent leaching by increasing the porosity of the shale.

(d) Leaching and uranium recovery

Countercurrent leaching using H_2SO_4 at 90°C (maintained by sparged steam). A recovery of 70% of the uranium is envisaged.

Pressure filtration is envisaged as being necessary owing to poor settling characteristics.

Leach solution is proposed to pass through a sand filter prior to processing in a SX plant using N-benzyl amine and stripping with sodium carbonate solution.

7. URANIFEROUS LIGNITES

Substantial deposits of uranium-containing lignites occur in several places in the United States of America.

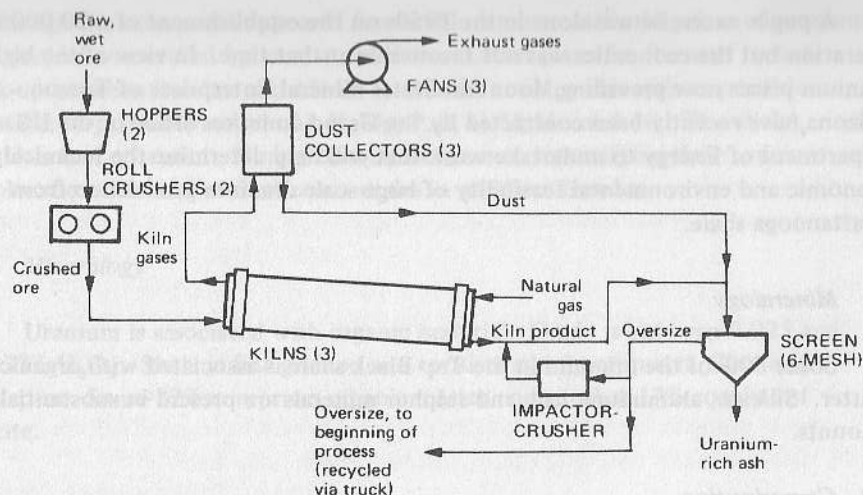


FIG.28. Lignite calcination flowsheet.

7.1. USA: Dakota, Montana, Wyoming

Uraniferous coal-type deposits appear generally to be restricted to a single seam, usually the highest of the local sequence with access to overlying source material, or at least connected with it by means of permeable strata. Some examples include 1.5 m thick seams in the Fort Union formations in South Dakota, beds of between 0.3 and 2.4 m thickness in Montana and uranium-bearing coal in the Red Desert area of Wyoming, USA.

For several years uranium was recovered from North Dakota lignites. The average feed grade was 4.2 kg/t U_3O_8 and 35% moisture; molybdenum content was in the range 2.5 to 10.0 kg/t and was a significant contribution to the favourable economics. The drying and calcination of the raw material produced an ash of which a typical assay was 6.8 kg/t U_3O_8 and was amenable to direct treatment in acid leach facilities when diluted with conventional uranium ore pulp. Direct processing of either the lignite or the ash alone was unsuccessful owing to the high organic contamination which rendered the solution unsuitable for the subsequent solvent extraction process. Uranium recovery in the calcination step was typically 93%. A typical calcination flowsheet is shown in Fig.28.

Although the subject of bench and pilot-scale studies, commercial processing of coals has been restricted to the addition of uraniumiferous lignite ash to conventional sandstone feed material as at Susquehanna-Weston. Uraniferous lignites require special handling and extraction technology and processing costs are not competitive with those for similar grade ore in existing uranium production areas.

(a) Mineralogy

No simple correlation exists between a coal's uranium content and any single petrographic component. South Dakota samples richest in uranium generally contain large amounts of attrital humic matter. In North Dakota visible uranium minerals are normally limited to semi-lustrous material in the weathered surface zone and to narrow vitrain bands in the richest portions of the deposits. Phosphate types are the most common. Among these, meta-autunite, sabugalite, and saleeite have been positively identified. The lignite may also contain yellow-green jarosite stains, which can be mistaken for uranium mineralization.

In South Dakota the higher-grade lignite beds contain molybdenum as jordisite and ilsemanite. In North Dakota the predominant gangue minerals are quartz, unidentified clay minerals, jarosite, limonite, pyrite, calcite and gypsum. Analcite spheroids, found in many of the South Dakota deposits, are characteristically absent. Concretions composed primarily of gypsum, iron pyrite and fragments of lignite occur in at least two unweathered deposits. No relationship has been observed between these minerals and the amount of uranium. Green iron-molybdenum minerals, one of which has been identified as ferromolybdenite, occur in intimate association with secondary uranium minerals in several of the higher-grade deposits. Lime content normally diminished inward from the lignite outcrop and appears to bear no consistent relationship to the amount of uranium.

(b) Comminution

Apart from crushing to effect a coal-shale separation, further liberation of the uranium-containing material is best accomplished by roasting and/or combustion of the carbonaceous material.

(c) Preconcentration or beneficiation

Assuming that the uranium is entirely contained in the coaly material then coal-washing processes such as jigging, dense media separation and flotation would apply.

(d) Leaching and uranium recovery

The possibilities include: (1) Leaching the coal; and (2) leaching an ash product.

In the latter case, both acid (H_2SO_4) and alkaline ($\text{Na}_2\text{CO}_3 + \text{NaHCO}_3$) leach processes have been investigated.

The presence of organic material causes the pulp to become gelatinous during leaching. Sodium bicarbonate additions in the presence of organic material cause foam formation.

Acid-leach liquors generally contain high levels of ferric iron thus precluding the use of alkyl phosphate solvents without a prior solution reduction stage. Organic material can be removed by using activated carbon (which can, in turn, be eluted using NaOH solution).

8. CALCRETES

8.1. Australia — Yeelirrie

The Yeelirrie uranium deposit in Western Australia occurs in sediments comprising sand, clay and calcrete in the Yilgarn Block, 65 km southeast of Wiluna. It is confined to a NNW-SSE-trending channelway which parallels green-stone belts in the region. There are undoubtedly other areas in the world where the favourable geology and climatic conditions needed to form calcrete-type uranium deposits are likely to have existed. Such deposits have reportedly been discovered in the Somalia Republic and in Namibia (South West Africa).

Western Mining announced their Yeelirrie discovery in 1972 and have conducted intensive investigations on the development of a process to treat the calcrete ore. Recently they announced plans to build a pilot plant to confirm design parameters and operating characteristics for a full-scale plant. The estimated cost of the programme, together with concurrent environmental studies and other development work, is around Aust. \$13 million. The construction of a 2500–3000 t U plant by 1980/81 is envisaged by Western Mining.

(a) Mineralogy

Uranium mineralization exists over a strike length of 45 km and an average width of 3 km. Precipitation of porous limestone (calcrete) has modified the original sand and clay stream-channel fill to varying depths and uranium occurs as thin coatings of carnotite in cavities and voids in the calcrete. The uranium grade varies, but the average for the whole deposit is approximately 0.15% U_3O_8 .

The gangue and host rock consists principally of quartz, dolomite, calcite, chert and feldspar with localized occurrences of gypsum, celestite, and clays.

(b) Comminution

The ore will probably be crushed in impact crushers to about minus 20 mm then dried and roasted in rotary kilns. The roasted ore will be quenched in a spiral classifier and the underflow fed to the grinding circuit, which will probably consist of rod and ball mills in series, grinding the ore to 80% minus 48 mesh.

(c) Pretreatment

Considerable attention will be paid to selectively mining and blending to produce a uniform feed material to the plant with as constant a head-grade as possible. The deposit is extremely shallow and occurs in roughly horizontal layers. Mining will probably be with scrapers and bulldozer-rippers. Once the overburden has been removed the exposed orebody will be closely drilled and radiometrically sampled before mining. The trucks from the mine will be monitored radiometrically to allow graded stockpiles to be prepared.

(d) Leaching

The high calcite content of the ore precludes any consideration of acid leaching. Alkaline leaching with sodium carbonate and bicarbonate solutions will be used.

(e) Solid-liquid separation

The main metallurgical problem is reportedly the poor solid-liquid characteristics of the leached pulp, owing to the clays present in the ore. Test work has shown that low-temperature roasting of the ore before leaching results in the destruction of the clay lattice by dehydration and leads to improved solid-liquid separation. A technique using pelletization followed by percolation leaching has been described that could also be applicable to solving this problem.

(f) Concentration/purification

No information has been published on the recovery circuit. Clearly the presence of vanadium (in the carnotite mineral) will have strong influence on the choice of process. Possibilities are direct precipitation of a crude yellowcake containing about 5% vanadium (that would have to be treated prior to acceptance by refineries) or the use of continuous ion-exchange columns for separation of uranium and vanadium, in which case a high-purity yellowcake could be produced directly from the eluate. However, it is believed that the local water supply may be too high in chloride to permit ion-exchange recovery.

9. URANIUM RECOVERY FROM UNCONVENTIONAL SOURCES

9.1. Copper leach liquors

Some United States mine operators have measured 1 to 12 ppm U_3O_8 in copper dump leaching solutions; with recirculation these latter concentrations may be increased to 40 ppm U_3O_8 . It is interesting to note that the US Bureau of Mines have on record a cement copper assay of 3100 ppm U_3O_8 . Until recently there was little effort made to recover uranium from this source. At present one commercial recovery plant is being commissioned in the USA, and test work is under way at other locations.

The process for recovering uranium from copper dump solutions involves passing leach liquors, discharged from the copper cementation stage, through ion-exchange units. When the resin becomes loaded with uranium it is eluted with strong sulphuric acid, and the elution liquor is processed in a solvent extraction circuit. The uranium from the concentrated solvent extraction strip liquor is precipitated as yellowcake. It is anticipated that the operating costs to extract uranium from copper dump leach liquors will be over US \$ 50/kg of U_3O_8 recovered.

Recently in a similar application at a United States location, test work and preliminary engineering studies were made on an Eluex/Bufflex process to recover the uranium in clarified solutions from leaching of oxide ores. It is not yet known whether the process will go into commercial operation.

Development work has also proceeded on the recovery of uranium from leach liquors prior to copper extraction.

9.2. Phosphoric acid

Extensive research into the recovery of uranium from phosphates has been under way for a number of years. Despite considerable efforts no process has been found that will selectively extract uranium from phosphate rock. Alkaline leaching methods are completely ineffective and acid leaching results in dissolving the uranium in approximately the same proportion as the dissolving of the phosphate rock. For these reasons all procedures for uranium recovery are based on by-product recovery from phosphoric acid.

The uranium content of the phosphoric acid is typically 150–175 ppm U_3O_8 . One plant is in operation in Florida, and two other facilities, one located in Louisiana the other in Florida, are scheduled to come on stream in 1978. In addition, two companies have recently completed designs of processing plants for Florida locations.

Three basic processing steps are involved in recovering uranium from this source.

- (1) Pre-treatment of 30% phosphoric acid to remove suspended solids and to adjust its redox potential;
- (2) Two successive stages of solvent extraction to separate the uranium from the phosphoric acid and then to concentrate it;
- (3) Precipitation of the uranium from the second-stage solvent extraction strip solution.

The main problems experienced in the pilot-plant operations were the formation of emulsions in solvent extraction which have led to high solvent losses, interference by iron in loading uranium on the solvent, and problems due to high temperatures and the corrosive nature of the high-fluoride-content phosphoric acid.

There are indications that the process for extraction of uranium from phosphoric acid is costly in terms of capital and operating requirements. Capital costs are reported to be in the range of US \$ 80–110/kg of annual U_3O_8 production with operating costs of the order of US \$ 44/kg U_3O_8 produced.

9.3. Sea-water

The uranium concentration in sea-water is probably stationary at 0.0033 ppm U_3O_8 ; it is estimated that the contribution from rivers, probably 27 000 t annually, is balanced by the deposition rate on the ocean floors. The dissolved uranium, which occurs as a tricarboxylate complex in the world's oceans, has been estimated to be more than 4×10^9 t.

In the early 1950s the United Kingdom Atomic Energy Authority started a programme of investigations into the recovery of uranium from sea-water. This was prompted by what appeared a national commitment to the development of nuclear power generation and a lack of domestic uranium ore reserves.

Following extensive laboratory testing of organic and inorganic materials, granular titanium hydroxide, used in conjunction with a sodium carbonate eluant, was found to be the most successful absorbent.

The initial test programme was terminated before procedures were optimized. Considerable research and development work will be required before a practical recovery process is obtained. The major difficulties to be resolved include:

- (1) The form of the uranium, which is strongly complexed and at extreme dilution; it exists in the presence of relatively high concentrations of what may be interfering ions;
- (2) The chemical and physical properties of the absorbent;

- (3) Type of eluant and effect of concentration and contact time;
- (4) Design and engineering of practical facilities to handle extremely high volumes of sea-water through filtration and extraction stages.

The Agency of Natural Resources and Energy in Japan have recently announced a two-year research programme to extract uranium from sea-water. Several mining, chemical and engineering companies are to be involved in the project. The experimental facility will be designed to recover 10 kg U/a; the absorbent is based on titanium oxide.

At this preliminary stage, where many key design features have not yet been proven or evaluated, no useful estimate of the economics can be presented.

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PART III

SUMMARY TABLES ON SPECIFIC URANIUM DEPOSITS AND PROCESSING PLANTS

Most of the examples given in the 76 Summary Tables at the end of this section are drawn from the major current and future uranium-producing countries (USA, Canada, South Africa, Namibia (South West Africa), Niger, France and Australia). However, examples have also been included of deposits with interesting characteristics from a further 12 countries (Mexico, Argentina, Brazil, Central African Empire, Gabon, Malagasy Republic, Austria, Denmark, Portugal, Spain, Sweden, and Yugoslavia).

The starting point for the selection of these examples was the December 1977 issue of the OECD-NEA/IAEA publication Uranium Resources, Production, and Demand. Altogether 49 countries were listed in the section of this book which summarizes the national submissions provided by government organizations responsible for the control of nuclear raw materials in their respective countries. In many of the 49 countries, specific deposits have yet to be delineated by further exploration drilling. In others, little information has been published on the mineralogy and amenability to processing of the deposits on which their resource figures are based. For these reasons 30 countries were excluded for the purposes of the Summary Tables. (As more information becomes available on uranium deposits and processing plants in these countries, additional examples will be added to the list for inclusion in future updates of the present report.)

1. EXPLANATION OF NUMBERING SYSTEM

1.1. Numbering of countries

The 49 countries included in the above book were grouped according to the following five regions:

1. NORTH AND CENTRAL AMERICA
2. SOUTH AMERICA
3. AFRICA AND THE MIDDLE EAST
4. WESTERN AND CENTRAL EUROPE
5. AUSTRALASIA AND THE FAR EAST

The countries in each region were then listed alphabetically and given a three-digit number as shown below.

Region 1: NORTH AND CENTRAL AMERICA

- 101 CANADA
- 102 MEXICO
- 103 UNITED STATES OF AMERICA

Region 2: SOUTH AMERICA

- 201 ARGENTINA
- 202 BOLIVIA
- 203 BRAZIL
- 204 CHILE
- 205 COLOMBIA
- 206 ECUADOR
- 207 PERU
- 208 URUGUAY

Region 3: AFRICA AND THE MIDDLE EAST

- 301 ALGERIA
- 302 CAMEROUN
- 303 CENTRAL AFRICAN EMPIRE
- 304 EGYPT
- 305 GHANA
- 306 IRAN
- 307 JORDAN
- 308 LIBERIA
- 309 MALAGASY REPUBLIC
- 310 MOROCCO
- 311 NAMIBIA (South West Africa)
- 312 NIGER
- 313 SOMALIA
- 314 SOUTH AFRICA
- 315 SUDAN
- 316 TURKEY
- 317 ZAIRE

Region 4: WESTERN AND CENTRAL EUROPE

- 401 AUSTRIA
- 402 DENMARK
- 403 FINLAND
- 404 FRANCE
- 405 GERMANY, Fed. Rep. of
- 406 GREECE
- 407 ITALY
- 408 PORTUGAL
- 409 SPAIN
- 410 SWEDEN
- 411 SWITZERLAND
- 412 UNITED KINGDOM
- 413 YUGOSLAVIA

Region 5: AUSTRALASIA AND THE FAR EAST

- 501 AUSTRALIA
- 502 INDIA
- 503 JAPAN
- 504 KOREA
- 505 NEW ZEALAND
- 506 PHILIPPINES
- 507 SRI LANKA
- 508 THAILAND

For convenient reference the 49 countries are shown, with the corresponding numbers, in Table A.

The list of 19 countries for which Summary Tables were included is shown in Table B.

1.2. Numbering of specific examples of deposits/mines/mills

As far as possible a standardized system of nomenclature was adopted for the deposits/mines/mills included in the Summary Tables. Thus, SOHIO-L BAR refers to the mill near Grants, New Mexico, that is operated by Sohio Petroleum Company and is often referred to as the L-Bar uranium mill. To pinpoint the

Text cont. on page 171

TABLE A. ALPHABETICAL LIST OF COUNTRIES INCLUDED IN
"URANIUM RESOURCES, PRODUCTION AND DEMAND"

Country		Country	
Algeria	(301)	Liberia	(308)
Argentina	(201)	Malagasy Republic	(309)
Australia	(501)	Mexico	(102)
Austria	(401)	Morocco	(310)
Bolivia	(202)	Namibia (SW Africa)	(311)
Brazil	(203)	New Zealand	(505)
Canada	(101)	Niger	(312)
Cameroun	(302)	Peru	(207)
Central African Empire	(303)	Philippines	(506)
Chile	(204)	Portugal	(408)
Colombia	(205)	Somalia	(313)
Denmark	(402)	South Africa	(314)
Ecuador	(206)	Spain	(409)
Egypt	(304)	Sri Lanka	(507)
Finland	(403)	Sudan	(315)
France	(404)	Sweden	(410)
Germany F.R.	(405)	Switzerland	(411)
Gabon	(305)	Thailand	(508)
Greece	(406)	Turkey	(316)
India	(502)	United Kingdom	(412)
Iran	(306)	United States of America	(103)
Italy	(407)	Uruguay	(208)
Japan	(503)	Yugoslavia	(413)
Jordan	(307)	Zaire	(317)
Korea	(504)		

TABLE B. LIST OF COUNTRIES FOR WHICH SUMMARY
TABLES WERE INCLUDED

Country	Number of examples
101 Canada	9
102 Mexico	1
103 USA	20
201 Argentina	1
203 Brazil	1
303 Central African Empire	1
305 Gabon	1
309 Malagasy Republic	1
311 Namibia (South West Africa)	1
312 Niger	4
314 South Africa	12
401 Austria	1
402 Denmark (Greenland)	1
404 France	5
408 Portugal	1
409 Spain	7
410 Sweden	1
413 Yugoslavia	1
501 Australia	7
TOTAL	76

TABLE C. LIST OF DEPOSITS/MINES/MILLS INCLUDED

Number	Name	Uranium area	State or province	Country
1-1	Amok-Cluff Lake	Carswell Structure	Saskatchewan	Canada
101-2	Denison	Elliot Lake	Ontario	Canada
101-3	Eldorado-Beaverlodge	North of Lake Athabasca	Saskatchewan	Canada
101-4	Gulf-Rabbit Lake	Wollaston Lake Fold Belt	Saskatchewan	Canada
101-5	Kerr Addison-Agnew Lake	Agnew Lake	Ontario	Canada
101-6	Madawaska-Bancroft	Bancroft	Ontario	Canada
101-7	Rio Algom-Quirke	Elliot Lake	Ontario	Canada
101-8	Rio Algom-Panel	Elliot Lake	Ontario	Canada
101-9	Preston Mines-Stanleigh	Elliot Lake	Ontario	Canada
102-1	El Nopal-Villa Aldama	Chihuahua	Chihuahua	Mexico
103-1	Anaconda-Bluewater	Grants Mineral Belt	New Mexico	USA
103-2	Atlas-Moab	Big Indian District	Utah	USA
103-3	Conoco-Falls City	South Texas	Texas	USA
103-4	Cotter-Canon City	Front Range	Colorado	USA
103-5	Dawn-Ford	Spokane	Washington	USA
103-6	Exxon-Powder River	Powder River Basin	Wyoming	USA
103-7	Federal American-Gas Hills	Gas Hills	Wyoming	USA
103-8	Kerr McGee-Grants	Grants Mineral Belt	New Mexico	USA
103-9	Pathfinder-Gas Hills	Gas Hills	Wyoming	USA
103-10	Pathfinder-Shirley Basin	Shirley Basin	Wyoming	USA
103-11	Rio Algom-La Sal	Big Indian District	Utah	USA
103-12	Sohio-L Bar	Grants Mineral Belt	New Mexico	USA
103-13	Union Carbide-Uravan	Uravan Mineral Belt	Colorado	USA
103-14	Union Carbide-Natrona	Gas Hills	Wyoming	USA
103-15	Union Carbide-Palangana	South Texas	Texas	USA
103-16	United Nuclear-Homestake	Grants Mineral Belt	New Mexico	USA
103-14	Union Carbide-Natrona	Gas Hills	Wyoming	USA
103-15	Union Carbide-Palangana	South Texas	Texas	USA
103-16	United Nuclear-Homestake	Grants Mineral Belt	New Mexico	USA
103-17	US Steel-Clay West	South Texas	Texas	USA
103-18	US Steel-Burns Ranch	South Texas	Texas	USA
103-19	Western Nuclear-Jeffrey City	Gas Hills	Wyoming	USA
103-20	Ranchers-Naturita	Uravan Mineral Belt	Colorado	USA

TABLE C (cont.)

Number	Name	Uranium area	State or province	Country
201-1	CNEA-Sierra Pintada	Sierra Pintada-Malargue	Mendoza	Argentina
203-1	Nuclebras-Pocos de Caldas	Minas Gerais	Minas Gerais	Brazil
303-1	Alusuisse-Bakouma	Bakouma	Mbomou	Central African Empire
305-1	Comuf-Mounana	Franceville	Haut-Ogooue	Gabon
309-1	Omnis-Fort Dauphin	Fort Dauphin	—	Malagasy Republic
311-1	Rio Tinto-Rössing	Swakopmund	—	Namibia (South West Africa)
312-1	Somair-Arlit	Agades	Azbine	Niger
312-2	Cominak-Akouta	Agades	Azbine	Niger
312-3	Imouranen	Agades	Azbine	Niger
312-4	Arni	Agades	Azbine	Niger
314-1	Anglo-Vaal Reefs	Klerksdorp	Transvaal	South Africa
314-2	Anglo-Western Deep Levels	West Witwatersrand	Transvaal	South Africa
314-3	Anglo-Ergo	East Rand Basin	Transvaal	South Africa
314-4	Anglo-President Brand	Free State Goldfields	Orange Free State	South Africa
314-5	Anglovaal-Hartebeestfontein	Klerksdorp	Transvaal	South Africa
314-6	General Mining-Buffelsfontein	Klerksdorp	Transvaal	South Africa
314-7	General Mining-W.R. Consolidated	Central and near West Rand	Transvaal	South Africa
314-8	Goldfields-West Driefontein	West Witwatersrand	Transvaal	South Africa
314-9	JCI-Randfontein	Central and near West Rand	Transvaal	South Africa

TABLE C (cont.)

Number	Name	Uranium area	State or province	Country
314-10	Rand Mines-Blyvooritzicht	West Witwatersrand	Transvaal	South Africa
314-11	Rand Mines-Harmony	Free State Goldfields	Orange Free State	South Africa
314-12	Rio Tinto-Palabora	Phalaborwa	Transvaal	South Africa
401-1	S.U.B.-Forstau	Radstadt	Salzburg	Austria
402-1	Risø - Kvanefjeld	Ilimaussaq	Greenland	Denmark
404-1	Simo-Bessines	Crouzille	Limousin	France
404-2	Simo-Saint Priest	Forez	Lyonnais	France
404-3	Simo-L'Ecarpière	Vendée	Poitou	France
404-4	Pouk-St.Pierre le Cantal	Massif Central	Massif Central	France
404-5	CFMU-Lozère	Herauld	Languedoc	France
404-6	Cogéma-Lodève	Herauld	Languedoc	France
408-1	Enu-Nisa	Nisa	Alto Alehtejo	Portugal
409-1	Enusa-Fe	Saelices el Chico	Salamanca	Spain
409-2	Jen-El Pedregal	La Haba	Badajoz	Spain
409-3	Jen-Los Ratones, Jen-Andujar	Albala	Caceres	Spain
409-4	Enusa-Mazarete	Molina de Aragón	Guadalajara	Spain
409-5	Fesa-Huelva	Phosphatic ores	Huelva	Spain
409-6	Enusa-Canidad	Villavieja de Yeltes	Salamanca	Spain
410-1	Lkab-Ranstad	Billingen	Vastergötland	Sweden
413-1	Boris Kidrič-Zirovsi Vrh	Northwest Yugoslavia	—	Yugoslavia
501-1	Ranger-Jabiru	Alligator Rivers	Northern Territory	Australia
501-2	Noranda-Koongarra	Alligator Rivers	Northern Territory	Australia
501-3	Pancontinental-Jabiluka	Alligator Rivers	Northern Territory	Australia

TABLE C (cont.)

Number	Name	Uranium area	State or province	Country
501-4	Queensland Mines-Nabarlek	Alligator Rivers	Northern Territory	Australia
501-5	Rio Tinto-Mary Kathleen	Mt Isa-Cloncurry	Queensland	Australia
501-6	Western Mining-Yeelirrie	Yilgarn Block	Western Australia	Australia
501-7	Western Nuclear-Beverley	Frome Embayment	South Australia	Australia

location of the deposit or operation more clearly in geological terms, the so-called "uranium area", in this instance the Grants Mineral Belt, is shown in the Tables, in addition to the State (New Mexico) and the country (USA). Once each of the examples was thus named they were listed alphabetically under each country and numbered in sequence as shown in Table C.

2. EXPLANATION OF DETAILS UNDER EACH HEADING

2.1. Owner/operator

The correct company designations are given here.

2.2. State of exploitation

An idea of the progress made towards the exploitation of the particular ore deposit is given under this heading in the Summary Tables. One or other of the following stages are usually applicable:

(a) *Evaluation*: The evaluation stage applies from the discovery of the uranium occurrence, through exploration drilling, mineralogical and ore processing investigations, pilot plant test work on bulk samples, pre-feasibility study, etc. to the point where the decision is made, in principle, to go ahead with mining and the erection of an ore processing plant.

(b) *Environmental impact*: This implies that the evaluation stage is complete but the go-ahead cannot be given because the authorities are still considering the environmental impact statement.

(c) *Construction/commissioning*: This implies that the mine and/or the ore processing plant is still being designed, constructed or commissioned.

(d) *Production*: This implies that the mine/mill is up to design capacity and that routine production of yellowcake is being undertaken.

(e) *Expansion/reactivation*: This category refers to old operations that have been on care and maintenance but are being brought back into production, or existing operations that are undergoing an expansion programme. In both cases the implication is that additional plant is being designed, erected and commissioned to cope with the reactivation or expansion.

(f) *Mine/mill shut-down*: Where difficult uranium ores were treated in the past by unusual methods that may still be of interest, the historical data is included in the tables and this heading is then allocated.

2.3. Grade

Because the grade of the ore being delivered to processing plants invariably fluctuates over a period of time as the mining operation gets into new areas of the orebody, it is preferable to quote a grade range rather than a specific average grade figure. For this reason, and to preserve confidentiality, one of the following grade ranges were chosen to indicate the approximate grade of each deposit.

% U ₃ O ₈	
< 0.05	0.50 – 1.00
0.05 – 0.10	1.00 – 2.00
0.10 – 0.15	2.00 – 3.00
0.15 – 0.20	3.00 – 5.00
0.20 – 0.30	5.00 – 10.00
0.30 – 0.40	> 10.00
0.40 – 0.50	Confidential/not known

2.4. Production

Because the actual throughput of processing plants does not necessarily coincide with the design capacity, and for reasons of confidentiality, one of the following production ranges was chosen to indicate the approximate production rate in tonnes of ore per day (330 days per annum) and also in tonnes U per annum.

(a) Tonnes ore per day:

Not in production yet	
< 500	8 000 – 9 000
500 – 1000	9 000 – 10 000
1000 – 2000	10 000 – 15 000
2000 – 3000	15 000 – 20 000
3000 – 4000	20 000 – 30 000
4000 – 5000	30 000 – 40 000
5000 – 6000	40 000 – 50 000
6000 – 7000	> 50 000 –
7000 – 8000	Confidential/not known

(b) Tonnes U per annum:

Not in production yet	
< 100	2000 – 3000
100 – 500	3000 – 4000
500 – 1000	4000 – 5000
1000 – 2000	> 5000
	Confidential/not known

2.5. Deposit ore type

The ore type in the Ruzicka classification that corresponds nearest to that of the deposit under consideration was chosen. Where it was felt appropriate, additional remarks on the type of ore and/or deposit were included.

2.6. Uraniferous minerals

Only the uranium minerals of major importance were listed. Where appropriate, additional remarks on the nature and distribution of the uranium minerals were included.

2.7. Other minerals

Accessory and gangue minerals that could have an influence on the selection of the process flowsheet or the economics were particularly noted.

2.8. Mining

Here an indication is given as to whether the type of mining being used or planned will be underground, open-pit, strip mining or solution mining (wells). Where a residue dump is being reclaimed an indication was given as to whether hydraulic or mechanical methods are being used or planned.

2.9. Sorting/blending

One or other of the following possible situations may apply here:

(a) *Separation of different ores at mine or mill in stockpiles:* No attempt is made to separate different ores at mine or mill (other than cutoff grade considerations); different grade ores stockpiled separately at mine or mill using radiometric properties; and ore from different mines stored separately at mill.

(b) *Blending at mill:* Different ores processed separately in campaigns in same plant; separate circuits for treating different ores; certain ores go through preparation stage (e.g. roasting) before joining main stream to mill; blending of all ores to produce continuous uniform feed to mill; and ore from mine directly to mill with minimal blending.

(c) *Sorting at mill:* No sorting; hand sorting; radiometric sorting; photometric sorting.

2.10. Preconcentration

Preconcentration to upgrade the ore (leaving a discardable tailing) or to split the ore into two streams for separate treatment is remarked upon here, where applicable.

2.11. Crushing

The crushing configuration (primary, secondary, tertiary) in various combinations and the type of crusher equipment are described here.

2.12. Milling

The milling configuration (number of stages, whether open or closed circuit) and the type of mill or mills are described.

2.13. Degree of comminution

To get an idea of the fineness of grind achieved by the milling circuit, the screen size (Tyler mesh) through which 80% of the solid particles in the feed to the leach pass is quoted. In addition, the screen size through which 50% of the solid particles in the same stream pass, is given as a rough indication of size distribution.

2.14. Leach type

The leach circuit may be either acid or alkaline (carbonate) while the leach method or equipment may be one of the following:

- Air-agitated pachucas
- Mechanical agitators
- Trommels (revolving drums)
- Autoclaves
- Heap leaching
- In-situ leaching
- Solution mining.

2.15. Oxidant/temperature

The oxidant may be one or other of the following:

- No oxidant added
- Manganese dioxide
- Sodium chlorate
- Ferric sulphate
- Oxygen (from air)
- Oxygen (from oxygen plant)
- Hydrogen peroxide
- Biological oxidation with bacteria
- Solution containing copper and ammonium ions.

Because leach temperatures vary from time to time it was decided to select one of the temperature ranges below rather than to quote a specific temperature:

Ambient temperature (°C)

35 – 45	90 – 95
45 – 55	95 – 100
55 – 65	100 – 120
65 – 75	120 – 150
75 – 80	150 – 200
80 – 85	200 – 250
85 – 90	> 250

The temperature ranges above 80°C are for pressure leach systems.

2.16. Leach pulp density

The leach pulp density expressed in percentage solids may vary from time to time and again, one of the ranges below is specified rather than a single figure:

Leach pulp density (% solids)

Not applicable (e.g. heap leaching)

< 20	50 – 55
20 – 25	55 – 60
25 – 30	60 – 65
30 – 35	65 – 70
35 – 40	70 – 75
40 – 45	> 75
45 – 50	

2.17. Solid-liquid separation

Here the number of stages of solid-liquid separation are quoted with the type of equipment used. One or more of the following may apply:

Rotary drum vacuum filters
Horizontal belt vacuum filters
Tilting pan vacuum filters
CCD in conventional thickeners
CCD in hi-rate thickeners (e.g. Enviro-Clear)
Sand/slimes separation in hydrocyclones
Classification in rake classifiers

Classification in spiral classifiers

Classification in drag classifiers

Rotoscoops

No solid-liquid separation equipment (e.g. heap leaching and resin-in-pulp).

2.18. Clarification

Here the equipment and clarification medium as well as the solids content of the clarified pregnant solution is noted. Possibilities are shown below.

Equipment

Leaf
Downflow columns
Upflow columns
Pressure filters
Tanks
Thickener

Clarification medium

Filter cloth
Sand
Anthracite coal/activated carbon
Precoat material

Solids content of pregnant solution

Clear solution	< 20 ppm
Solution with low solids content	20 – 1000 ppm
Solution with moderate solids content	1000 ppm – 1%
Dilute pulps	1 – 10%
Medium pulps	10 – 20%
Thick pulps	> 20%

2.19. Concentration/purification

The four main possibilities are

Ion exchange (IX) only
Solvent extraction (SX) only

Combination of IX and SX (Eluex, Bufflex)
Recovery and product precipitation combined (e.g. as for alkaline leach circuits).

The type of equipment used in the recovery step may be one or more of the following:

Fixed-bed ion exchange
Fixed-bed ion exchange (resin transfer type)
NIMCIX continuous ion exchange
Porter CIX
Davy Powergas CIX
Himsley CIX
Higgins/Chemsep CIX
Pechiney Ugine Kuhlmann CIX
Basket RIP
Screen-mix RIP
Mixer-settlers SX
Pulse columns SX
Kerr-McGee single-stage upflow ion exchange

2.20. Configuration (of recovery process)

The number of stages and their description are noted under this heading. The examples are as follows:

Two columns for absorption and one column for elution (3 fixed-bed columns);
Two columns for absorption, one column for backwash, one column for elution (fixed-bed IX resin);
Single column for absorption, single column for elution (upflow column CIX).
20 vessels on absorption, two on backwash, two on elution (basket RIP).

2.21. Resin/solvent

One or other of the following possibilities may apply:

Conventional strong-base IX resin
Heavy strong-base IX resin (S.G. > 1.2)
Conventional weak-base IX resin

Heavy weak-base IX resin (S.G. > 1.2)
Tertiary amine solvent
Tributyl phosphate solvent.

2.22. Elution/stripping

The basic elution or stripping medium, whether nitrate, sulphate, chloride, carbonate, or ammonia, is mentioned. Where appropriate, more details are given, e.g. "10% sulphuric acid elution" for Eluex/Bufflex systems or "3-stage pH increase with ammonium sulphate recycle" for SX systems.

2.23. Precipitation type

The basic type, the equipment and the reagents may be one or other of the following:

Basic type

Intermediate impurity precipitation from eluate or strip solution before uranium precipitation;
Precipitation of uranium directly from eluate or strip solution;
Precipitation of uranium directly from leach liquors.

Equipment

Conventional stirred tanks;
Unconventional equipment (specify).

Reagents for precipitation

Calcium hydroxide (lime)
Calcium carbonate
Magnesia
Ammonia gas
Ammonium hydroxide
Sodium hydroxide
Hydrogen peroxide

2.24. Drying/calcination

One of the following may apply:

- Rotary type
- Belt (conveyor) type (e.g. Proctor Swartz)
- Screw type
- Rotating-table type
- Single hearth
- Multiple hearth (e.g. Skinner roaster).

2.25. Product

Final products are either termed yellowcake or uranium oxide concentrates. Where the yellowcake is magnesium or sodium diuranate this is specified in parentheses. Where hydrogen peroxide is used for precipitation the product is termed uranium peroxide. The method of despatch of the final product is usually in drums, except in South Africa where ammonium diuranate slurry is despatched in road tankers from each mine to a central calcining and drumming plant.

2.26. Unusual features

Provision is made here for a short note on unusual features relating to the deposit, the mine or the processing plant.

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SUMMARY TABLES

1 0 1 - 1	AMOK - CLUFF LAKE	Saskatchewan (Carswell Structure)	CANADA
Owner/Operator	Mokta + Pechiney + Cogema + C.F.M.U. / Amok (Canada) Limited.		
Stage of Exploitation	Mining of D orebody scheduled to start in 1980. Expansion after 1982.		
Grade (Range)	D Orebody 5-10% U_3O_8 . Claude and N orebodies 0.5 - 1.0% U_3O_8 .		
Production (Range)	< 500 t/d initially expanding to 1000 - 2000 t/d (1000 - 2000 t U/a)		
Deposit Ore Type	Vein - on or close to unconformity		
Uraniferous Minerals	D orebody contains pitchblende and uraninite. In the N orebody pitchblende and coffinite occurs along shear zones. In the Claude orebody uraninite and coffinite are found in flat-lying shears in quartzitic gneiss.		
Other Minerals	Gold is an accessory mineral in the D orebody (0.5 - 1.0 ounce per ton) which also contains small amounts of selenium, cobalt, bismuth, lead and various selenides and tellurides. All three orebodies contain varying amounts of pyrite, galena, pyrrhotite and chalcopyrite. The N and Claude orebodies contain garnet, marcasite, cordierite and sillimanite.		
Mining	Open pit. Exposed bench to be covered with non-radioactive overburden for radiation shielding, before proceeding to next bench. Selective mining to be practised.		
Sorting/Blending	No sorting. No blending		
Preconcentration	Gravity concentration is possible on the D orebody to produce a concentrate with a uranium concentration of 45% U_3O_8 .		
Crushing	Primary and secondary shorthead crushers		
Grinding	Single stage rod milling in closed circuit with classifiers		
Degree of Comminution	80% minus 48 mesh (50% minus 65 mesh)		
Leach Type	Acid leach in mechanical agitators		
Oxidant/Temperature (Range)	Sodium chlorate. 45 - 55°C.		
Leach Pulp Density (Range)	60 - 70% solids decreasing sharply as uranium dissolution occurs.		
Remarks on Leaching			
Solids/Liquid Separation	Two stages of horizontal belt filters		
Clarification	No clarification		
Solids content of preg. soln	< 20 p.p.m.		
Concentration/Purification	Uranium to be precipitated directly from leach liquors without prior concentration or purification by either ion exchange or solvent extraction.		
Configuration (SX, IX)			
Resin /Solvent			
Elution /Stripping			
Precipitation Type	Magnesium diuranate precipitated using magnesia in stirred tanks.		
Drying/Calcination	Conveyor (tunnell type dryer).		
Product	Dried yellow cake (magnesium diuranate) in drums.		
Unusual Features	High Grade. Complex Gangue Mineralogy. Radiation shielding. Radon emissions. Remote location. Extreme climate. No IX or SX. Crude product.		

April 1979

1 0 1 - 2	DENISON	Ontario (Elliot Lake)	CANADA
Owner/Operator	Denison Mines Limited.		
Stage of Exploitation	6400 tpd mill being expanded to 10900 t/d by 1982, 13600 t/d possible by 1985.		
Grade (Range)	0.10 - 0.15% U_3O_8		
Production (Range)	10 000 - 15 000 t/d. 2000 - 3000 t U/a		
Deposit Ore Type	Conglomerate - pyritic quartz-pebble conglomerate of the Matinenda Formation.		
Uraniferous Minerals	Brennerite, uraninite and monazite		
Other Minerals	The matrix of the quartz-pebble conglomerates, in which the uranium mineralization occurs, consists principally of quartz, sericite, chlorite and pyrite. Yttrium is recovered as a by-product.		
Mining	Underground. Three variations of the room and pillar method are employed viz. Drift Jumbo, Longhole and Alimak.		
Sorting/Blending	No sorting. No blending		
Preconcentration	No preconcentration.		
Crushing	Jaw for primary, standard cone for secondary, shorthead cone for tertiary.		
Grinding	Two-stage grind with rod and pebble mills in closed circuit with rake classifiers.		
Degree of Comminution	60% minus 65 mesh (50% minus 200 mesh)		
Leach Type	Conventional acid leach in air agitated pachuca.		
Oxidant/Temperature (Range)	Sodium chlorate 65 - 75°C		
Leach Pulp Density (Range)	77% solids		
Remarks on Leaching	After grinding, dewatering in disc filters and repulping with royal barren solution, sulphuric acid is added in a conditioning tank prior to 40 hour leach in pachuca.		
Solids/Liquid Separation	Single stage hydrocyclones. Cyclone overflow to 3 stage CCD in conventional thickeners.		
Clarification	Cyclone underflow to 2 stages of rotary drum vacuum filters.		
Solids content of preg. soln	Leaf clarifiers. Filter cloth. Precast material.		
Concentration/Purification	Fixed-bed ion exchange (resin transfer type).		
Configuration (SX, IX)	Two stages of absorption, one backwash, one elution.		
Resin /Solvent	Conventional strong-base resin.		
Elution /Stripping	Nitrate elution.		
Precipitation Type	Intermediate impurity precipitation with CaO and NH_3 in pachuca, before uranium precipitation with NH_3 in stirred tanks		
Drying/Calcination	Conveyor type (Proctor Swartz)		
Product	Dried yellow cake in drums		
Unusual Features	None		

April 1978.

101-3	ELDORADO - BEAVER LODGE	Saskatchewan (North of Lake Athabasca)	CANADA
Owner/Operator	Crown Corporation of Federal Government / Eldorado Nuclear Limited.		
Stage of Exploitation	Expansion from 900 t/d to 1270 t/d by 1978 and 1630 t/d by 1979.		
Grade (Range)	0.15 - 0.25% U_3O_8		
Production (Range)	1000 - 2000 t/d 500 - 1000 t U/a		
Deposit Ore Type	Vein		
Uraniferous Minerals	Mainly pitchblende with minor uranophane.		
Other Minerals	Most of the ores contain calcite, quartz and minor pyrite. The Belgar, Nicholson and Martin Lake ores also contain copper sulphides, cobalt-nickel sulpharsenides, silver, gold and selenides. The Gunnar ore also contains chlorite, kaolinite, chalcocypite and galena. Carbonate ranges from 5 - 8% in feed to mill.		
Mining	Underground. To date ore has been treated from Eldorado's own properties. In future it will also act as a custom mill for ore from other mines in the area. Small number of open pit operations.		
Sorting/Blending	No selective mining. No blending		
Preconcentration	Flotation of classifier overflow. Sulphide concentrate to acid leach. Tailings to alkaline leach.		
Crushing	The ore is extremely hard with a Bond work index of 20. Run of mine ore is crushed underground to minus 15 cm.		
Grinding	The coarse ore is ground in a double-compartment semi-autogenous mill which operates in closed circuit with a cyclone and classifier in series. The latter, in turn, operates in closed circuit with two ball mills. Plant solution, containing 100g/l of dissolved carbonate and sulphate salts is used for milling.		
Degree of Comminution	Classifier overflow is 25% solids and 75% minus 200 mesh.		
Leach Type	Alkaline (carbonate) leach in mechanical agitators (air for emergencies).		
Oxidant/Temperature (Range)	Oxygen from O_2 plant (or emergency air). 90 - 95°C.		
Leach Pulp Density (Range)	40 - 50% solids		
Remarks on Leaching	Circuit incorporates a small batch leach plant for acid leaching of pyrite cleaner concentrate (H_2SO_4 + sodium chlorate). Acid pulp is filtered and both Fe and U_3O_8 precipitated with magnesium hydroxide. Crude cake is redissolved in carbonate leach circuit.		
Solids/Liquid Separation	Two-stage filtration with rotary drum vacuum filters (string discharge)		
Clarification	Leaf classifiers. Filter cloth. Precoat material (Supercol).		
Solids content of preg. soln.	< 20 p.p.m.		
Concentration/Purification	Precipitation of uranium directly from leach liquors. (Two-stage evaporation/steam stripping prior to precipitation and recycle of reagents via recarbonation step)		
Configuration (SX, IX)			
Resin /Solvent			
Elution /Stripping			
Precipitation Type	Sodium diuranate precipitated with sodium hydroxide in series of 5 conventional stirred tanks.		
Drying/Calcination	Screw type. (Holo-flite). 2 stage.		
Product	Dried yellow cake (sodium diuranate) in drums.		
Unusual Features	Complex gangue mineralogy. High Work Index. Flotation to split ore into two streams for separate treatment by acid and alkaline leaching. Remote location. Extreme climate.		

September 1979

101-4	GULF - RABBIT LAKE	Saskatchewan (Wollaston Lake Field Belt)	CANADA
Owner/Operator	Gulf Oil Corporation + Uranerz (U. Germany) / Gulf Minerals Canada Limited.		
Stage of Exploitation	Design capacity of 1800 t/d achieved late in 1978.		
Grade (Range)	0.30 - 0.40% U_3O_8 (erratic grades)		
Production (Range)	1000 - 2000 t/d 1000 - 2000 t U/a		
Deposit Ore Type	Vein		
Uraniferous Minerals	Pitchblende, carnotite and minor uranophane.		
Other Minerals	Hematite, calcite, dolomite, vughy quartz, chlorite and clay minerals. Minor amounts of pyrite, marcasite, chalcocypite and galena.		
Mining	Open pit.		
Sorting/Blending	Selective mining using radiometric monitoring of trucks. Simple blending in stockpiles. No sorting.		
Preconcentration	No preconcentration.		
Crushing	Jaw for primary crushing.		
Grinding	The ore is ground in two stages in an autogenous mill followed by a ball mill in closed circuit with cyclones.		
Degree of Comminution	50% minus 200-mesh		
Leach Type	Acid leach in mechanical agitators.		
Oxidant/Temperature (Range)	Sodium chlorate / 65 - 75°C		
Leach Pulp Density (Range)	50% solids		
Remarks on Leaching	Cyclone overflow passes directly to the acid leach circuit i.e. there is no dewatering stage.		
Solids/Liquid Separation	5 stage CCD in conventional thickeners. Underflow density achieved is 55% solids. Overflow contains 75 p.p.m. solids.		
Clarification	Primary clarification in a thickener. Secondary clarification in downflow clarifiers		
Solids content of preg. soln.	< 20 p.p.m.		
Concentration/Purification	Solvent extraction in mixer-settlers.		
Configuration (SX, IX)	4 extraction, 4 stripping, no scrubbing.		
Resin /Solvent	Tertiary amine solvent.		
Elution /Stripping	Ammonia strip (pH increased in 3 stages, ammonium sulphate recycle)		
Precipitation Type	Precipitation of uranium with NH_3 from strip liquor in stirred tanks.		
Drying/Calcination	Hearth type.		
Product	Yellow cake in drums.		
Unusual Features	No town nearby. All workers flown in and out alternate weeks.		

April 1979

101-5	KERR ADDISON - AGNEW LAKE	Ontario (Agnew Lake)	CANADA
Owner/Operator	Subsidiary of Kerr Addison (43% Noranda Mines) / Agnew Lake Mines Limited.		
Stage of Exploitation	Commissioning of plant almost completed - design capacity by 1979.		
Grade (Range)	0.05 - 0.10% U_3O_8		
Production (Range)	t/d not applicable. 100 - 500 t U/a		
Deposit Ore Type	Conglomerate - (pyritic quartz-pebble conglomerate of the Matinenda Formation)		
Uraniferous Minerals	Uranothorite, monazite (Th/U ratio is approx. 3)		
Other Minerals	Quartz, Sericite, pyrite, pyrrhotite. Rich in rare earths.		
Mining	Underground mining coupled with in-situ leaching in stopes.		
Sorting/Blending	Not applicable		
Preconcentration	Not applicable		
Crushing	No crushing on underground ore. Primary crushing for surface stockpile.		
Grinding	No milling		
Degree of Comminution	Not applicable		
Leach Type	Underground acid in-situ leaching and acid heap leaching on surface.		
Oxidant/Temperature (Range)	Biological oxidation with bacteria (with aeration). Ambient temperature.		
Leach Pulp Density (Range)	Not applicable		
Remarks on Leaching	Most of leaching done in stopes underground but because of increased space required for broken ore and acid spray system about 30 - 45% of ore must be hoisted for surface heap leaching.		
Solids/Liquid Separation	Not applicable.		
Clarification	Pregnant solutions are collected in ponds lined with Hypalon membranes. Ore piled on P.V.C. membrane base. Downflow sand clarifiers between CIX and Eluex SX.		
Solids content of preg. soln	Pregnant solution to CIX contains 20 - 1000 p.p.m. solids		
Concentration/Purification	Himalay CIX coupled with solvent extraction (Eluex/Bufflex circuit).		
Configuration (SX, IX)	Single stage absorption, backwash, elution (3 sets) for CIX 3 extraction, 1 scrub, 4 strip, 2 conditioning stages for SX		
Resin /Solvent	Conventional strong-base resin. Tertiary amine solvent.		
Elution /Stripping	10% H_2SO_4 elution, ammonia strip for SX.		
Precipitation Type	NH_3 gas and NH_4OH (ammonia recovery system) for uranium precipitation in conventional stirred tanks (3 in series)		
Drying/Calcination	Multiple hearth		
Product	Dried yellow cake in drums		
Unusual Features	Novel underground mining coupled with in-situ leaching in stopes. Heap leaching on surface. Processing plant must deal with impurities introduced underground in leach solutions and all ore brought to surface is heap leached.		

September 1976

101-6	MADAWASKA - BANCROFT	Ontario (Bancroft)	CANADA
Owner/Operator	Can-Ped. Res. Corporation + Con. Faraday Ltd. / Madawaska Mines Limited.		
Stage of Exploitation	Plant built in 1957. Closed 1964. Rehabilitation of mine/mill 1976. Full capacity 1977.		
Grade (Range)	0.05 - 0.10 % U_3O_8		
Production (Range)	1000 - 2000 t/d 100 - 500 t U/a		
Deposit Ore Type	Pegmatite (in highly metamorphosed pre-cambrian sediments)		
Uraniferous Minerals	Uranium is present as uraninite and uranothorite as disseminated, discrete crystals within the pegmatite. Minor betafite, allanite, zircon and fergusonite occur.		
Other Minerals	Quartz, pyroxene, amphibole, biotite, muscovite		
Mining	Underground		
Sorting/Blending	No sorting. No blending		
Preconcentration	No preconcentration.		
Crushing	Jaw for primary and shorthead for secondary crushing.		
Grinding	Two-stage grind with rod and pebble mills in closed circuit with rake classifiers.		
Degree of Comminution	80% minus 100 mesh (50% minus 200 mesh)		
Leach Type	Acid leach in air-agitated pechuoas		
Oxidant/Temperature (Range)	Sodium chlorate. Ambient temperature		
Leach Pulp Density (Range)	60 - 70%.		
Remarks on Leaching	Dewatering in thickeners prior to leach. 38 hours residence time at pH 1.5 start to 1.0 discharge		
Solids/Liquid Separation	Two-stage rotary drum vacuum filters - string discharge		
Clarification	Rotary drum precoat filter followed by pressure sand clarifier.		
Solids content of preg. soln	20 - 1000 p.p.m.		
Concentration/Purification	Fixed-bed ion exchange		
Configuration (SX, IX)	Three stages absorption, single stage elution.		
Resin /Solvent	Conventional strong-base resin.		
Elution /Stripping	Sodium chloride acidified with H_2SO_4 for elution.		
Precipitation Type	Batch precipitation in stirred tanks using magnesia		
Drying/Calcination	Multiple Hearth Dryer.		
Product	Dried yellow cake (magnesium diuranate) in drums (some calcining).		
Unusual Features	Relatively low grade pegmatite ore being processed by conventional flowsheet.		

April 1977

101-7	RIO ALGOM - QUIRKE	Ontario (Elliot Lake)	CANADA
Owner/Operator	Rio Algom Limited.		
Stage of Exploitation	Recent expansion from 4080 t/d to 6350 t/d.		
Grade (Range)	0.10 - 0.15% U_3O_8 .		
Production (Range)	8000 - 7000 t/d 1000 - 2000 t U/a		
Deposit Ore Type	Conglomerate (pyritic quartz-pebble conglomerate of the Matinenda Formation)		
Uraniferous Minerals	Uraninite, brannerite, monazite.		
Other Minerals	Quartz, sericite, chlorite, pyrite.		
Mining	Underground		
Sorting/Blending	No sorting. No blending.		
Preconcentration	No preconcentration.		
Crushing	Jaw for primary (underground), standard cone for secondary, shorthead cone for tertiary.		
Grinding	Two stage grind in closed circuit with classification using rod and pebble mills.		
Degree of Comminution	80% minus 65 mesh (50% minus 200 mesh)		
Leach Type	Conventional acid leach in air-agitated pachuca.		
Oxidant/Temperature (Range)	No oxidant added (O_2 from air only). 65 - 75°C.		
Leach Pulp Density (Range)	70 - 80% solids		
Remarks on Leaching	Dewatering in disc filters prior to leach. 40 hours residence time.		
Solids/Liquid Separation	Four stage hydrocyclones send relief circuit - washed sands join washed slimes from CCD circuit at neutralization pachuca then to tailings dam. Cyclone overflow to five-stage CCD in conventional thickeners. Sixth stage of CCD recently added as Enviro-clear thickeners.		
Clarification	Leaf clarifiers. Filter cloth. Precoat of diatomaceous earth.		
Solids content of preg. soln	< 20 p.p.m.		
Concentration/Purification	Fixed-bed ion exchange		
Configuration (SX, IX)	Two stages absorption, one stage elution.		
Resin /Solvent	Conventional strong-base resin.		
Elution /Stripping	Nitrate elution.		
Precipitation Type	Pachuca for Fe precipitation with CaO and NH_3 . Conventional stirred tanks for uranium precipitation with NH_3 .		
Drying/Calcination	Conveyor type (Proctor Swartz)		
Product	Dried yellowcake in drums.		
Unusual Features	Enviro-Clear thickeners being assessed.		

April 1978

101-8	RIO ALGOM - PAREL	Ontario (Elliot Lake)	CANADA
Owner/Operator	Rio Tinto Zinc Corporation / Rio Algom Limited.		
Stage of Exploitation	Reactivation and increase in capacity from 2720 t/d to 3000 t/d. Start up late 1979.		
Grade (Range)	0.10 - 0.15% U_3O_8 .		
Production (Range)	3000 - 4000 t/d 500 - 1000 t U/a		
Deposit Ore Type	Conglomerate (pyritic quartz-pebble conglomerate of the Matinenda Formation)		
Uraniferous Minerals	Uraninite, brannerite, monazite.		
Other Minerals	Quartz, sericite, chlorite, pyrite.		
Mining	Underground.		
Sorting/Blending	No sorting. No blending.		
Preconcentration	No preconcentration.		
Crushing	Jaw for primary (underground), standard cone for secondary, shorthead cone for tertiary.		
Grinding	Two stage grind in closed circuit with classifiers using rod and pebble mills.		
Degree of Comminution	80% minus 65 mesh (50% minus 200 mesh)		
Leach Type	Conventional acid leach in air-agitated pachuca.		
Oxidant/Temperature (Range)	No oxidant added (only oxygen from air) 65 - 75°C.		
Leach Pulp Density (Range)	70 - 80% solids.		
Remarks on Leaching	Dewatering in disc filters prior to leach - 40 hours residence time.		
Solids/Liquid Separation	Single CCD stage in conventional thickeners followed by two-stage filtration in rotary drum vacuum filters.		
Clarification	Leaf clarifiers. Filter cloth. Precoat diatomaceous earth		
Solids content of preg. soln	< 20 p.p.m.		
Concentration/Purification	Fixed-bed ion exchange (resin transfer type)		
Configuration (SX, IX)	Three stages absorption, one backwash, three elution.		
Resin /Solvent	Conventional strong base resin.		
Elution /Stripping	10% sulphuric acid elution.		
Precipitation Type	Impurity precipitation from eluate in mechanical agitators using calcium carbonate and MgO followed by uranium precipitation in conventional stirred tanks using magnesia.		
Drying/Calcination	Conveyor type dryer (Proctor Swartz)		
Product	Dried yellow cake (magnesium diuranate) in drums.		
Unusual Features	First example where 10% H_2SO_4 elution of strong-base resin is not coupled with SX i.e. Buffles/Elux configuration. Two stage precipitation preferred.		

April 1979

101-9	PRESTON MINES - STANLEIGH	Ontario (Elliot Lake)	CANADA
Owner/Operator	Rio Tinto Zinc Corporation / Rio Algom Limited.		
Stage of Exploitation	To be reactivated and expanded from 2720 t/d to 4500 t/d. Start up 1983/4		
Grade (Range)	0.05 - 0.10% U_3O_8 .		
Production (Range)	4000 - 5000 t/d 500 - 1000 t U/a.		
Deposit Ore Type	Conglomerate (pyritic quartz-pebble conglomerate of the Matinanda Formation)		
Uraniferous Minerals	Uraninite, brannerite, monazite.		
Other Minerals	Quartz, sericite, chlorite, pyrite.		
Mining	Underground		
Sorting/Blending	No sorting. No blending.		
Preconcentration	No preconcentration.		
Crushing	Jaw for primary crushing (underground)- cone for secondary crushing.		
Grinding	Two-stage grind in closed circuit with classification. Autogenous mill with pebble ports following pebble mill. Excess pebbles crushed.		
Degree of Comminution	80% minus 65 mesh (50% minus 200 mesh)		
Leach Type	Acid leach in air agitated pachuca.		
Oxidant/Temperature (Range)	No oxidant added (only oxygen from air). 65 - 75°C		
Leach Pulp Density (Range)	> 70% solids		
Remarks on Leaching	Dewatering in disc filters prior to leach. 40 hours retention time. Pre-aeration after grinding.		
Solids/Liquid Separation	Single CCD stage in conventional thickeners followed by two-stage filtration in rotary drum vacuum filters. (horizontal belt filters being assessed as an alternative).		
Clarification	No clarification		
Solids content of preg. soln	20 - 1000 p.p.m.		
Concentration/Purification	Continuous ion exchange (probably Himsley CIX)		
Configuration (SX, IX)	One absorption column, one backwash, one elution.		
Resin /Solvent	Conventional strong-base ion exchange resin.		
Elution /Stripping	10% sulphuric acid elution.		
Precipitation Type	Impurity precipitation from eluate in air and mechanical agitators using calcium carbonate and MgO, followed by uranium precipitation with magnesia in conventional stirred tanks.		
Drying/Calcination	Multiple hearth dryer.		
Product	Dried yellow cake (magnesium diuranate) in drums.		
Unusual Features	Note : Data above on proposed flowsheet are tentative and still subject to change		

April 1978

102-1	EL NOPAL - VILLA ALDARA	Chihuahua	MEXICO
Owner/Operator	Instituto Nacional de Energia Nuclear (INEN)		
Stage of Exploitation	Pilot plant evaluation stage.		
Grade (Range)	0.20 - 0.30% U_3O_8		
Production (Range)	< 500 t/d < 100 t U/a		
Deposit Ore Type			
Uraniferous Minerals	Mineralization occurs in a "chimney" of rhyolite and tuff as disseminated primary and secondary oxides and fracture fillings.		
Other Minerals			
Mining			
Sorting/Blending			
Preconcentration	No preconcentration.		
Crushing			
Grinding			
Degree of Comminution			
Leach Type	Acid heap leaching		
Oxidant/Temperature (Range)	No oxidant required. Ambient temperature.		
Leach Pulp Density (Range)	Not applicable.		
Remarks on Leaching	Choice was between heap (percolation) leach with 70% efficiency and conventional leach with 90% efficiency. Even though grade is relatively high, heap leaching was chosen.		
Solids/Liquid Separation	Not applicable		
Clarification			
Solids content of preg. soln	< 20 p.p.m.		
Concentration/Purification	Solvent extraction in mixer-settlers.		
Configuration (SX, IX)	Four stages of extraction, one stage water scrub, three stage strip.		
Resin /Solvent	Tertiary amine solvent.		
Elution /Stripping	250 g/l ammonium sulphate and 100 g/l ammonium chloride.		
Precipitation Type			
Drying/Calcination			
Product			
Unusual Features			

April 1978

103-1	ANACONDA - BLUEWATER	New Mexico (Granta Minera) Belt	U.S.A.
Owner/Operator	Atlantic Richfield Co. (ARCO) / Anaconda Company.		
Stage of Exploitation	Expansion in 1978 from 3000 t/d to 8000 t/d. Flowsheet changed.		
Grade (Range)	0.15 - 0.20% U_3O_8 .		
Production (Range)	5000 - 8000 t/d. 2000 - 3000 t U/a.		
Deposit Ore Type	Sandstone. (Jackpile) Uranium occurs in "stacked" layers in conglomeratic, feldspathic sandstone lenses, east of Mt. Taylor.		
Uraniferous Minerals	Uranium occurs as coffinite and minor uraninite and as unidentified black, friable, uranyl complexes. Near the surface autunite, tyuyamunite, meta-tyuyamunite, andersonite, and uranophane occur as coatings or small fracture fillings.		
Other Minerals	Pyrite and non-commercial quantities of selenium, vanadium and molybdenum occur.		
Mining	In 1977 the mill received ore from 1 source with underground and open-pit operations.		
Sorting/Blending	Different grade ores stockpiled separately after radiometric monitoring of trucks. Simple blending in dumping pockets. No sorting.		
Preconcentration	No preconcentration.		
Crushing	Primary crushing at mine.		
Grinding	Semi-autogenous milling of ore in closed circuit with cyclones.		
Degree of Comminution	80% minus 35 mesh (50% minus 65 - 100 mesh)		
Leach Type	Acid leach.		
Oxidant/Temperature (Range)	Manganese dioxide as oxidant. 55 - 85°C.		
Leach Pulp Density (Range)	50 - 55% solids		
Remarks on Leaching			
Solids/Liquid Separation	Seven stage CCD in Enviro-Clear thickeners (30ft diameter)		
Clarification	Pressure sand clarifiers		
Solids content of preg. soln	< 20 p.p.m. solids		
Concentration/Purification	Solvent extraction in mixer-settlers.		
Configuration (SX, IX)	Four extraction stages, no scrubbing, four stripping stages.		
Resin/Solvent	Tertiary amine solvent.		
Elution/Stripping	Chloride stripping.		
Precipitation Type	Two stage precipitation in conventional stirred tanks using magnesia.		
Drying/Calcination	Rotary dryer.		
Product	Dried yellow cake (magnesium diuranate) in drums.		
Unusual Features	One of the first large-scale applications of Enviro-Clear thickeners in uranium circuit		

December 1978.

103-2	ATLAS - ROAD	Utah (Big Indian District)	U.S.A.
Owner/Operator	Atlas Corporation / Atlas Minerals		
Stage of Exploitation	Production (RIP acid circuit 1950 changed to alkaline 1961 and SX acid circuit added, 1967)		
Grade (Range)	0.10 - 0.15% U_3O_8		
Production (Range)	1000 - 1500 t/d 200 - 800 t U/a		
Deposit Ore Type	Sandstone - uranium occurs in fluvial, feldspathic, conglomeratic sandstone units on the southwest plane of the Lisbon Valley anticline.		
Uraniferous Minerals	Uraninite		
Other Minerals	Vanadium as oxides and silicates, calcite, pyrite, chalcocopyrite, bornite. Vanadium recovered as by-product.		
Mining	Underground. Plant served by approximately 60 mines.		
Sorting/Blending	None		
Preconcentration	Flotation used at one time. Now discontinued.		
Crushing	Jaw crusher for primary, cone crusher for secondary.		
Grinding	Two-stage ball milling closed circuit with classifier and cyclone. Identical arrangement for both acid and alkaline circuits.		
Degree of Comminution	Alkaline circuit - 60 to 65% minus 200 mesh. Acid circuit - 50% minus 85 mesh.		
Leach Type	Alkaline - Pressure leach in autoclaves. Acid - Leach in mechanical agitators.		
Oxidant/Temperature (Range)	Alkaline - air / 100 - 200°C Acid - sodium chlorate when required / 75 - 80°C.		
Leach Pulp Density (Range)	50 - 55% solids.		
Remarks on Leaching	High lime, low pyrite ores treated by alkaline leaching. High vanadium, low lime ores treated by acid leaching. (Flotation and copper extraction no longer practised.)		
Solids/Liquid Separation	Alkaline - 3 stage filtration Acid - 4 stage CCD plus single-stage filtration.		
Clarification	Alkaline - pressure sand filters Acid - no clarification		
Solids content of preg. soln	< 20 p.p.m. (alkaline circuit).		
Concentration/Purification	Alkaline - caustic precipitation. Acid - solvent extraction.		
Configuration (SX, IX)	Uranium SX plant contains 4 extraction (amine) and 2 stripping (soda-ash) stages. Vanadium SX plant treating uranium SX raffinate extracts vanadium as a cationic complex.		
Resin/Solvent	Tertiary Amine solvent for U SX. Di-2-ethyl-hexylphosphoric acid (EHPA) for V SX		
Elution/Stripping	Stripping is done with carbonate for U SX, and with 25% H_2SO_4 for V SX.		
Precipitation Type	Alkaline caustic precipitation followed by redissolution with sulphuric acid and combination with acid circuit acidified strip solution, followed by final H_2O_2 precipitation.		
Drying/Calcination	Multiple hearth furnace for uranium. 3-hearth dryer and fusion furnace for vanadium.		
Product	Uranium peroxide is calcined to uranium oxide concentrate for despatch in drums. V_2O_5 flakes produced as a by-product.		
Unusual Features	Plant serves about 60 mines with wide variety of ore types. Wide variety of process techniques employed. Peroxide precipitation from uranium. By-product recovery of vanadium.		

December

103-3	CONOCO - FALLS CITY	South Texas	U.S.A.
Owner/Operator	Pioneer Nuclear Corp. + Continental Oil Company / Conoco		
Stage of Exploitation	Routine production (start-up 1972).		
Grade (Range)	0.05 - 0.15% U_3O_8		
Production (Range)	2500 - 3500 t/d 500 - 1000 t U/a		
Deposit Ore Type	Sandstone - Uranium occurs in narrow lenticular bodies in poorly cemented feldspathic sandstones (Fordilla). Mines as much as 35 miles away from the Falls City - Conquista Creek area.		
Uraniferous Minerals	Uraninite and Coffinite.		
Other Minerals	Bentonites. Some ores contain lignites. Generally the ores treated are lower in carbonate content than other deposits to the southwest.		
Mining	Overburden removed by scrapers. In 1977 mill received ore from about 10 open-pit operations.		
Sorting/Blending	No sorting. Stockpiles at mine according to grade.		
Preconcentration	No preconcentration.		
Crushing	Primary crushing with impact type crushers.		
Grinding	Single stage open circuit rod mills.		
Degree of Comminution	80% minus 35-mesh (50% minus 65 - 100 mesh)		
Leach Type	Acid leach in mechanical agitators.		
Oxidant/Temperature (Range)	Sodium chlorate / 55 - 65°C.		
Leach Pulp Density (Range)	45 - 50% solids		
Remarks on Leaching	Heat for leaching derived from semi-artesian well-water (65°C)		
Solids/Liquid Separation	5 stage CCD in conventional thickeners.		
Clarification	Downflow sand columns.		
Solids content of preg. soln	< 20 p.p.m. solids		
Concentration/Purification	Solvent Extraction in mixer-settlers.		
Configuration (SX, IX)	4 stage extraction, 4 stage stripping.		
Resin /Solvent	Tertiary amine solvent.		
Elution /Stripping	Acidified ammonium chloride for stripping.		
Precipitation Type	Direct precipitation from strip solution with NH_3 gas in conventional stirred tanks (2 in series)		
Drying/Calcination	Multiple hearth (3-hearth BSP dryer)		
Product	Dried yellowcake in drums.		
Unusual Features	Relatively low grade ores from a number of sources being treated by conventional technology.		

December 1978.

103-4	COTTER - CANON CITY		Colorado (Front Range)	U.S.A.
Owner/Operator	Commonwealth Edison / Cotter Corporation			
Stage of Exploitation	New mill being installed to treat 1200 t/d. Old mill (450 t/d) to treat tailings.			
Grade (Range)	Mined ores 0.20 - 0.30% U_3O_8 . Tailings dam 0.025% U_3O_8 .			
Production (Range)	1000 - 1500 t/d. 200 -1000 t U/a.			
Deposit Ore Type	Sandstone ores from western slope of Colorado Plateau (75%). Vein type ore from Schwartzwalder mine (25%).			
Uraniferous Minerals	Sandstone ore similar to Ureva Mineral Belt - see 103-13 Vein type ores - pitchblende and minor coffinite occurs as coatings with secondary minerals in fractures within the oxidized zone.			
Other Minerals	Sandstone ores - see 103-13 Vein type ores - Pyrite, quartz, ankerite, minor lead, zinc, copper sulphides.			
Mining	In 1977 mill received ore from about 10 mines. Underground mining at Schwartzwalder.			
Sorting/Blending	Three ore bins for Western slope sandstone, Schwartzwalder and custom milled ores respectively. Blended feed produced containing approx. 75% sandstone ore.			
Preconcentration	None			
	OLD ALKALINE PLANT		NEW ACID PLANT	
Crushing	Primary jaw and secondary gyratory.		Western slope ores precrushed. Primary crush of Schwartzwalder onsite. All ore campaigns through secondary crusher on site.	
Grinding	Ball mill in closed circuit with spiral classifier.		Open circuit rod mill followed by ball mill	
Degree of Comminution	70% minus 200-mesh		97% minus 65-mesh (design figure).	
Leach Type	Alkaline carbonate leach in air-agitated pachuca and autoclaves.		Acid leach in mechanical agitators + pressure acid leach in Sheritt-Gordon type autoclaves.	
Oxidant/Temperature (Range)	Air as oxidant. 76°C pachuca 110°C autoclaves		Air as oxidant. 115°C in autoclaves.	
Leach Pulp Density (Range)	40% solids		50% solids	
Remarks on Leaching	30 hours in pachuca + 1 hour in autoclaves (latter at 3-4 atmospheres). Alkaline autoclave FRP lined.		5 atmospheres pressure in acid autoclaves, brick-lined impellers, and let-down valves are titanium.	
Solids/Liquid Separation	3-stage countercurrent rotary drum filtration.		9-stage CCD.	
Clarification	Sand clarification.		Thickener-type clarifier followed by pressure downflow filtration.	
Solids content of preg. soln	< 20 p.p.m.		< 20 p.p.m.	
Concentration/Purification			Solvent extraction for uranium (USX) + vanadium (VSX)	
Configuration (SX, IX)	Direct precipitation from alkaline leach solutions.		USX 4 extraction, 4 strip, 2 regen. VSX 4 extraction, 6 strip.	
Resin /Solvent			USX Alamine 304 (low Mo extraction) VSX Alamine 336	
Elution /Stripping			USX NH_3 stripping with $(NH_4)_2SO_4$ solution regen VSX same as USX (alternatively soda ash)	
Precipitation Type	NaOH precip., redissolution in H_2SO_4 and H_2O_2 precipitation.		Activated carbon to remove Mo. ADU precip.	
Drying/Calcination	Rotating hearth dryer (gas fired)		Hearth Dryer.	
Product	Uranium peroxide calcined to oxide.		Dried yellowcake in drums.	
Unusual Features	A variety of unusual processes are included in the overall flowsheet. Molybdenum extraction plant treats soda ash regen and activated carbon as well as molybdenum catalysts from oil refineries.			

December 1978

December 1978

103-5	DAWN - FORD	Washington (Spokane)	U.S.A.
Owner/Operator	Newmont Mining Corp. (51%) + Midnight Mining (49%) / Dawn Mining Company.		
Stage of Exploitation	Routine production. Start-up 1957. On stand-by 1965. Reactivated 1975.		
Grade (Range)	0.10 - 0.15% U_3O_8 .		
Production (Range)	◀ 500 t/d. 100 - 500 t U/a.		
Deposit Ore Type	Vein type - Orebodies are discordant, lenticular and hosted at the contact of a dark-grey elaty argillite-phyllite within the Togo formation and a porphyritic quartz monzonite.		
Uraniferous Minerals	Uranium occurs as black, sooty uraninite and coffinite. Pitchblende specimens are fairly common. The shallow orebodies were oxidized and contained autunite, meta-autinite, uranophane, and phosphuranylite with iron oxide coatings.		
Other Minerals	Pyrite, marcasite, pyrrhotite, molybdenite, quartz, calcite.		
Mining	Open pit.		
Sorting/Blending	No sorting. Stockpile blending.		
Preconcentration	No preconcentration.		
Crushing	Jaw for primary and impact for secondary.		
Grinding	Ball mills in closed circuit with cyclones.		
Degree of Comminution	50% minus 200-mesh		
Leach Type	Acid leach in mechanical agitators.		
Oxidant/Temperature (Range)	Sodium chlorate / 45 - 55°C.		
Leach Pulp Density (Range)	50% solids.		
Remarks on Leaching	Two-stage acid leach circuit. Milled pulp diluted with solution from second stage prior to first stage leaching. Thickener after first leach stage-overflow is pregnant solution, underflow goes to second leach stage where acid is added.		
Solids/Liquid Separation	5 stage CCD in conventional thickeners.		
Clarification	Downflow columns containing anthracite.		
Solids content of preg. soln	< 20 p.p.m. solids.		
Concentration/Purification	Fixed-bed ion exchange.		
Configuration (SX, IX)	3 columns for absorption, 1 column for elution.		
Resin /Solvent	Conventional strong-base resin.		
Elution /Stripping	Chloride elution.		
Precipitation Type	Impurity precipitation with lime followed by uranium precipitation with ammonia.		
Drying/Calcination	Proctor and Schwartz steam-heated dryer.		
Product	Dried yellowcake in drums.		
Unusual Features	Two-stage acid leach.		

December 1978.

103 - 6	EXXON - POWDER RIVER	Wyoming (Powder River Basin)	U.S.A.
Owner/Operator	Exxon Company		
Stage of Exploitation	2000 t/d in 1972 expanded to 3000 t/d without major construction programme.		
Grade (Range)	0.10 - 0.15% U_3O_8		
Production (Range)	2500 - 3500 t/d - 1000 - 1500 U/a.		
Deposit Ore Type	Sandstone - roll front orebodies along channel margins, tabular orebodies along permeability changes and elliptical orebodies within scour pockets. Fort Union Formation.		
Uraniferous Minerals	Coffinite and uraninite accompanied by carbonaceous debris and pyrite. Uranium occurs as thin sooty layers and spherical masses on detrital sandstone grains. The most prevalent of the near surface minerals are carnotite and meta-tyuyamunite and minor carbonates such as liebigite, bayleyite and zellerite.		
Other Minerals			
Mining	One open pit and one underground mine.		
Sorting/Blending	No sorting. Stockpile blending.		
Preconcentration	No preconcentration.		
Crushing	Open circuit impact crusher.		
Grinding	Open circuit rod mill.		
Degree of Comminution	80% minus 35-mesh (50% minus 65 - 100 mesh)		
Leach Type	Acid leach in mechanical agitators.		
Oxidant/Temperature (Range)	Sodium chlorate. Ambient temperature.		
Leach Pulp Density (Range)	50% solids.		
Remarks on Leaching	None		
Solids/Liquid Separation	5 stage CCD in conventional thickeners.		
Clarification	Pressure sand bed clarifiers.		
Solids content of preg. soln	< 20 p.p.m.		
Concentration/Purification	Solvent extraction in mixer-settlers. Upflow ion-exchange on tailings water.		
Configuration (SX, IX)	4 extraction, 4 stripping.		
Resin /Solvent	Tertiary amine solvent.		
Elution /Stripping	Sulphate stripping.		
Precipitation Type	Conventional precipitation with ammonia.		
Drying/Calcination	6 hearth roaster.		
Product	Dried yellowcake in drums.		
Unusual Features	Upflow ion exchange on tailings water.		

December 1978.

103-7	FEDERAL AMERICAN - GAS HILLS	Wyoming (Gas Hills)	U.S.A.
Owner/Operator	Federal Resources (30%) - American Nuclear Co. (20%) TVA (50%) / Federal American Partners.		
Stage of Exploitation	520 t/d in 1959. Expanded to 950 t/d with Eluex added to RIP in 1963. Further expansion 1978/79.		
Grade (Range)	0.10 - 0.15% U_3O_8 .		
Production (Range)	500 - 1000 t/d 100 - 500 t U/a.		
Deposit Ore Type	Sandstones. Yellowish to grey arkosic sandstone, interstratified with mudstones, and shales in the Wind River Formation. Roll type orebodies aligned in belts.		
Uraniferous Minerals	Earthy brown to black coatings and interstitial fillings of uranium and coffinite. Over 20 minor uranium minerals have been identified.		
Other Minerals	Black ores are enriched in selenium, molybdenum, and arsenic, which are not present in the barren rock. Selenium is abundant near the contact between the ore and the altered sandstone.		
Mining	In 1977 the mill received ore from one source. Open pit mining.		
Sorting/Blending	No sorting except for low, medium and high stockpiles at mine according to grade determined by radiometric measurements. Initial blending in mill feed stockpiles, further blending by drawing combined mill feed from coarse ore storage bins. Rotary kiln for drying ore when necessary.		
Preconcentration	No preconcentration.		
Crushing	Jaw crusher for primary, gyratory crusher for secondary.		
Grinding	Single stage ball mill in closed circuit with spiral classifier.		
Degree of Comminution	80% minus 35-mesh (50% minus 65 - 100 mesh)		
Leach Type	Acid leach in mechanical agitators (6 in series).		
Oxidant/Temperature (Range)	Sodium chlorate. 35 - 45°C.		
Leach Pulp Density (Range)	50 - 55% solids.		
Remarks on Leaching	As much acid as possible added to first agitator. Oxidant added to last two agitators to maintain high extraction. Part of uranium already dissolved has tendency to reprecipitate if e.m.f. drops below -390m.v. Electrodeless conductivity probe used for control of acid addition.		
Solids/Liquid Separation	Initial sand/slime separation in single stage 10 inch cyclones. Sands washed in 3 stage counter-current spiral classifier circuit. Overflow from cyclones and classifiers processed through counter-current circuit of 6 inch cyclones in a complex configuration.		
Clarification	No clarification.		
Solids content of preg. soln	Solids content of pregnant pulp to RIP is 12 - 15%.		
Concentration/Purification	Screen-mix resin-in-pulp. Eluex process (RIP + SX)		
Configuration (SX, IX)	RIP - 7 vessels for loading, 2 for washing, 11 for elution. SX - 4 extraction, 4 strip mixer-settlers.		
Resin /Solvent	RIP grade strong base resin. Tertiary amine solvent.		
Elution /Stripping	12 - 15% H_2SO_4 for elution. Controlled pH strip with ammonia.		
Precipitation Type	Conventional precipitation with ammonia.		
Drying/Calcination	6 Hearth Skinner roaster.		
Product	Calcined uranium oxide in drums.		
Unusual Features	Sand-slime separation and washing circuit, RIP circuit.		

December 1978.

103-8	KERR MCGEE - GRANTS	New Mexico (Grants Mine) Belt	U.S.A.
Owner/Operator	Kerr-McGee Nuclear Corporation		
Stage of Exploitation	Routine Production. Start up 1958.		
Grade (Range)	0.15 - 0.25% U_3O_8 .		
Production (Range)	5000 - 6000 t/d 3000 - 4000 t U/a.		
Deposit Ore Type	Sandstone - Uranium occurs in clusters of elongated pods (trend ore) and in "stacked" layers in feldspathic to arkosic sandstone lenses in the Westwater Canyon and Poison Canyon Members of the Morrison Formation. Both oxidized and unoxidized redistributed orebodies are abundant in proximity to carbonaceous debris in the arkosic fluvial sandstone.		
Uraniferous Minerals	Fine-grained coffinite is the main uranium mineral with minor uraninite. Oxidized zones contain secondary uranium minerals (of pre-mining interest only). Unidentified uranium probably present as urano-organic complexes.		
Other Minerals	Paramontrosite, pyrite, jordisite, ferroselite and marcasite.		
Mining	In 1977 the mill received ore from about 20 mines. Kerr-McGee own 9 underground mines in Ambrosia Lake area.		
Sorting/Blending	No sorting. No blending. Dumping pockets to keep ore separate.		
Preconcentration	No preconcentration		
Crushing	Jaw for primary, impact for secondary crushing.		
Grinding	Single stage open circuit rod mills.		
Degree of Comminution	80% minus 35 mesh (50% minus 65 - 100 mesh)		
Leach Type	Acid leach in mechanical agitators.		
Oxidant/Temperature (Range)	Sodium chlorate. 55 - 65°C.		
Leach Pulp Density (Range)	50 - 55% solids		
Remarks on Leaching			
Solids/Liquid Separation	Single stage hydrocyclones for sand/slime separation. Sands washed in 5 stage rake classifiers. Slimes washed in 8 stage CCD in conventional thickeners.		
Clarification	Pressure filters. Precoat material.		
Solids content of preg. soln	<20 p.p.m.		
Concentration/Purification	Solvent extraction in mixer settlers. (Upflow IX on mine waters)		
Configuration (SX, IX)	4 stages of extraction. no scrubbing, 4 stages of stripping.		
Resin /Solvent	Tertiary amine solvent. (strong-base resin for IX)		
Elution /Stripping	Chloride stripping for both SX and mine water IX)		
Precipitation Type	Conventional precipitation with NH_3 gas in stirred tanks (4 in series)		
Drying/Calcination	Conveyor belt dryer (Proctor-Schwartz)		
Product	Dried yellowcake in drums.		
Unusual Features	Largest uranium producer in the USA. Upflow IX for mine waters.		

December 1978

103-9	PATHFINDER - GAS HILLS	Wyoming (Gas Hills)	U.S.A.
Owner/Operator	Pathfinder Mines Corporation.		
Stage of Exploitation	1200 t/d in 1958. Being expanded to 2800 t/d by 1979		
Grade (Range)	0.15 - 0.20% U_3O_8 .		
Production (Range)	2000 - 3000 t/d. 500 - 1000 t U/a.		
Deposit Ore Type	Sandstone - Roll type orebodies aligned in belts in the Wind River Formation. Yellowish to gray arkosic sandstones, interstratified with mudstones and shales.		
Uraniferous Minerals	Urandinite and coffinite. Over 20 minor uranium minerals.		
Other Minerals	Selenium is abundant near the contact between the ore and the altered sandstone. Molybdenum and arsenic present.		
Mining	In 1977 the mill received ore from 2 sources. Open pit mining.		
Sorting/Blending	No sorting. Truckloads classified by radiometric measurement and stockpiled according to grade at mill. Blending from stockpiles.		
Preconcentration	No preconcentration.		
Crushing	None		
Grinding	Semi-autogenous grinding mill.		
Degree of Comminution	80% minus 35-mesh. (50% minus 65 - 100 mesh).		
Leach Type	Acid leach in mechanical agitators.		
Oxidant/Temperature (Range)	Sodium chlorate. 35°C.		
Leach Pulp Density (Range)	55 - 60% solids		
Remarks on Leaching			
Solids/Liquid Separation	DSM screen separates coarse sand for washing in 5 stage hydrocyclone circuit. DSM u/f to hydrocyclone. Cyclone o/f to RIP and u/f to 6 stage CCD in conventional thickeners.		
Clarification	Carbon bed clarifiers.		
Solids content of preg. soln	<20 p.p.m.		
Concentration/Purification	Fixed-bed ion exchange (resin transfer type). Eluex (IX + SX). Screen-mix RIP		
Configuration (SX, IX)	2 columns on absorption (3 at times), 1 column on backwash, resin transfer tank for elution. 4 extraction, 1 scrub, 3 strip for SX.		
Resin /Solvent	Conventional strong base resin. Tertiary amine solvent.		
Elution /Stripping	10% H_2SO_4 elution. Controlled pH ammonia strip.		
Precipitation Type	Conventional precipitation with NH_3 gas in single stage after Mo removal from strip solution with activated charcoal.		
Drying/Calcination	Conveyor belt dryer. (Proctor-Schwartz)		
Product	Dried yellowcake in drums.		
Unusual Features	December 1975.		

103-10	PATHFINDER - SHIRLEY BASIN	Wyoming (Shirley Basin)	U.S.A.
Owner/Operator	Pathfinder Mines Corporation.		
Stage of Exploitation	Routine production. Start-up 1971.		
Grade (Range)	0.15 - 0.20% U_3O_8 .		
Production (Range)	1000 - 2000 t/d. 500 - 1000 t U/a.		
Deposit Ore Type	Sandstone - Host rocks for large roll-type deposits are yellow to gray arkosic sandstones of Wind River Formation.		
Uraniferous Minerals	Urandinite in fracture fillings and coatings.		
Other Minerals	Pyrite, calcite, hematite and marcasite		
Mining	Ore treated from one source only. Open pit mining.		
Sorting/Blending	No sorting. Trucks monitored radiometrically. Stockpiles according to grade at mill. Stockpile blending aimed at constant feed to mill.		
Preconcentration	No preconcentration.		
Crushing	No crushing.		
Grinding	Single-step semi-autogenous Cascade mill.		
Degree of Comminution	80% minus 35 mesh (60% minus 65 - 100 mesh)		
Leach Type	Acid leach in air-agitated pachuca.		
Oxidant/Temperature (Range)	Sodium chlorate. 35 - 45°C.		
Leach Pulp Density (Range)	55 - 50% solids		
Remarks on Leaching	2 storage pachuca followed by 4 air-agitated leach pachuca.		
Solids/Liquid Separation	6 hydrocyclones (18 inch) for sand/slime separation and sand washing.		
Clarification	6 stage CCD in conventional thickeners for slimes washing.		
Solids content of preg. soln	4-20 p.p.m.		
Concentration/Purification	Fixed bed ion-exchange (resin moved out and back periodically to prevent channelling).		
Configuration (SX, IX)	3 columns on absorption, 1 column on elution.		
Resin /Solvent	Macroreticular weak-base ion exchange resin.		
Elution /Stripping	Sodium chloride elution.		
Precipitation Type	Precipitation of uranium directly from eluate with NH_3 gas in stirred tanks (2 stages).		
Drying/Calcination	Hearth Roaster		
Product	Dried yellowcake in drums.		
Unusual Features	Use of weak base resins and chloride elution eliminates need for Eluex (IX and SX)		

103-11	RIO ALGOM - LA SAL	Utah (Big Indian District)	U.S.A.
Owner/Operator	Rio Tinto Zinc Corporation / Rio Algom Corporation.		
Stage of Exploitation	1972 at 500 t/d. Minor modifications to increase to 750 t/d completed in 1975.		
Grade (Range)	0.20 - 0.25% U_3O_8 .		
Production (Range)	500 - 800 t/d. 100 - 500 t U/a.		
Deposit Ore Type	Sandstone - Grebodies are tabular, amoeba-shaped masses, concordant to the bedding in the lowest member of the Moss Back member of the Chinle Formation, predominantly a fluvialite calcareous, conglomeritic sandstone.		
Uraniferous Minerals	Uraninite fills pores and partly replaces sand grains and fossil wood fragments. Coffinite also present. Oxidized zones contain meta-tyuyamunite, carnosite and carnotite.		
Other Minerals	Vanadium minerals - pascoite, montrosite, doloresite, vanadium clay and hydronite. Mill heads average about 10% calcium carbonate (calcite).		
Mining	Underground. Grebody developed on a double entry system which outlined stope areas in 150 ft square blocks.		
Sorting/Blending	Each block is sampled around the perimeter and an average grade and thickness is assigned to allow flexibility in planning the blending of various grades to produce an optimum grade of feed to the mill.		
Preconcentration	No preconcentration.		
Crushing	Primary crushing in hammermill.		
Grinding	Single stage ball mill in closed circuit with cyclone.		
Degree of Comminution	80% minus 100 - mesh (99% minus 65 - mesh)		
Leach Type	Preleach (alkaline) in air-agitated pachuca followed by alkaline pressure leach in autoclaves.		
Oxidant/Temperature (Range)	Oxygen in compressed air. 75°C in pachuca. 120°C in autoclaves.		
Leach Pulp Density (Range)	50% solids.		
Remarks on Leaching	Milled pulp after thickening, passes to 3 pre-leach pachuca with 17 hours retention time, then to 2 parallel banks of 5 autoclaves where retention time is 7 hours.		
Solids/Liquid Separation	3 stages of filtration in rotary drum filters (string discharge)		
Clarification	Leaf clarifiers. Filter cloth.		
Solids content of preg. soln	<20 p.p.m.		
Concentration/Purification	Precipitation of uranium directly from leach liquors with sodium hydroxide. Seven tanks in series.		
Configuration (SX, IX)	-		
Resin /Solvent	-		
Elution /Stripping	-		
Precipitation Type	Sodium diuranate is either dried directly (if specifications allow it) or redissolved in H_2SO_4 followed by ammonium diuranate precipitation.		
Drying/Calcination	6-hearth dryer		
Product	Either sodium diuranate or ammonium diuranate.		
Unusual Features	At 2700ft Lisbon Mine is currently the deepest in the world for sandstone ores. December 1978.		

103-12	SOHIO - L. B. A. H.	New Mexico (Grants Mineral Belt)	U.S.A.
Owner/Operator	Sohio Petroleum & Reserve Oil and Minerals Corporation/ Sohio Petroleum Corporation.		
Stage of Exploitation	Routine production. Start up 1976.		
Grade (Range)	0.10 - 0.20% U_3O_8		
Production (Range)	1000 - 2000 t/d. 500 - 1000 t U/a.		
Deposit Ore Type	Sandstone - Uranium occurs in "stacked" layers in sandstone lenses.		
Uraniferous Minerals	Coffinite, uraninite and a unidentified black, friable uranyl complexes.		
Other Minerals	Pyrite and non-commercial quantities of selenium, vanadium and molybdenum.		
Mining	In 1977 mill received ore from about 5 sources. Open pit and underground.		
Sorting/Blending	No sorting. No blending.		
Preconcentration	No preconcentration.		
Crushing	No crushing.		
Grinding	Semi-autogenous cascade mill in closed circuit with cyclones.		
Degree of Comminution	80% minus 35 - mesh (50% minus 65 - 100 mesh, 20 - 30% minus 200 mesh).		
Leach Type	Acid leach in mechanical agitators.		
Oxidant/Temperature (Range)	Sodium chlorate. 50°C.		
Leach Pulp Density (Range)	45 - 50% solids.		
Remarks on Leaching	Mechanically agitated, flat-bottomed, RLMS leach tanks.		
Solids/Liquid Separation	6 stage CCD in conventional thickeners.		
Clarification	Pressure down-flow clarifiers. Anthracite beds.		
Solids content of preg. soln	< 20 p.p.m.		
Concentration/Purification	Solvent extraction in mixer-settlers (Pump-mix design)		
Configuration (SX, IX)	4 extraction, 4 strip.		
Resin /Solvent	Tertiary amine solvent.		
Elution /Stripping	Controlled pH ammonia strip with ammonium sulphate recycle.		
Precipitation Type	Conventional ammonia gas precipitation in 2 tanks in series.		
Drying/Calcination	Turbo-tray oil-fired dryer (Wysmont hearth dryer).		
Product	Dried yellowcake in drums.		
Unusual Features	Special attention given to the use of a variety of different linings in tanks etc. to meet different conditions. December 1978.		

103 - 13	UNION CARBIDE - URAVAN	Colorado (Uravan Mineral Belt)	U.S.A.
Owner/Operator	Union Carbide Corporation.		
Stage of Exploitation	Routine production. Present mill constructed 1955/56 to replace original built in 1936.		
Grade (Range)	0.15 - 0.20% U_3O_8 .		
Production (Range)	1000 - 1500 t/d 500 - 1000 t U/a.		
Deposit Ore Type	Sandstone - Uppermost carbonaceous sandstone unit of the Salt Wash member of the Morrison Formation and are generally concordant to the bedding of the host rock.		
Uraniferous Minerals	Until 1960 production largely from oxidized near-surface deposits. Now mostly unoxidized ores containing uraninite and coffinite.		
Other Minerals	Vanadium silicates and montroseite.		
Mining	In 1977 the mill received ^{from} about 60 mines Underground mines within 40 mile radius of plant.		
Sorting/Blending	No sorting. Ores from different mines stored separately at mill until sampled. Then certain amount of blending done.		
Preconcentration	No preconcentration.		
Crushing	Jaw crusher for primary crushing.		
Grinding	Single stage Aerofall mill (dry grinding) in closed circuit with dry cyclone and wet scrubber.		
Degree of Comminution	80% minus 35 - mesh (50% minus 65 - 100 mesh).		
Leach Type	Acid leach in mechanical agitators.		
Oxidant/Temperature (Range)	Sodium chlorate. 85 - 90°C		
Leach Pulp Density (Range)	50 - 80% solids.		
Remarks on Leaching	Two stage leach. Second stage uses hot, strong acid treatment to improve extraction of vanadium. Strong acid liquors are separated from second stage leached pulp in CCD circuit and partially neutralized in first stage by fresh dry ore.		
Solids/Liquid Separation	4 stage CCD in woodstave thickeners of conventional design.		
Clarification	Extra thickener.		
Solids content of preg. soln	<20 p.p.m.		
Concentration/Purification	Fixed bed ion exchange (Vanadium is reduced with SO_2 to tetravalent state in pregnant solution and therefore does not load on resin).		
Configuration (SX, IX)	3 columns on loading, 1 column on washing or stand-by, 1 column on elution.		
Resin /Solvent	Weak base resin.		
Elution /Stripping	Split elution technique. Chloride.		
Precipitation Type	Conventional NH_3 gas precipitation in 2 vessels in series.		
Drying/Calcination	Multiple hearth. (Skinner roaster)		
Product	Calcined uranium oxide concentrate in drums.		
Unusual Features	Plant in two sections built at different levels along the side of a hill. Custom mill for very large number of small mining operations. December 1978.		

103 - 14	UNION CARBIDE - NATRONA	Wyoming (Gas Hills)	U.S.A.
Owner/Operator	Union Carbide Corporation.		
Stage of Exploitation	Start up 1960. Eluex added subsequently.		
Grade (Range)	0.10 - 0.15% U_3O_8 .		
Production (Range)	1000 - 1500 t/d 100 - 500 t U/a.		
Deposit Ore Type	Sandstone - Roll type orebodies in Wind River Formation. Yellowish to gray arkosic sandstones interstratified with mudstones and shales.		
Uraniferous Minerals	Uraninite and coffinite. Over 20 minor uranium minerals.		
Other Minerals	Selenium, molybdenum and arsenic.		
Mining	Ore received from 5 mines in 1977. Open pit.		
Sorting/Blending	No sorting. Ore directly to mill with minimal blending except that ore high in Mo is diluted with low Mo ore.		
Preconcentration	No preconcentration.		
Crushing	Primary crushing only. Jaw or Hammermill at option of operators.		
Grinding	Single stage open circuit rod milling.		
Degree of Comminution	80% minus 35 - mesh (50% minus 65 - 100 mesh).		
Leach Type	Acid leach in mechanical agitators.		
Oxidant/Temperature (Range)	Sodium chlorate. 45 - 55°C.		
Leach Pulp Density (Range)	45 - 50% solids.		
Remarks on Leaching	Attempts have been made to treat 0.03% ore by underground in-situ and heap leaching. Recoveries poor. Heap leach piles tend to seal off with consequent low percolation rates.		
Solids/Liquid Separation	6 stages of countercurrent washing using a parallel arrangement of cyclones and spiral classifier to separate sands and slimes. Slimes dewatered in conventional thickener to RIP feed.		
Clarification	No clarification		
Solids content of preg. soln	10 - 20% solids		
Concentration/Purification	Screen mix RIP (with air agitation). Eluex circuit (RIP and SX).		
Configuration (SX, IX)	RIP - 7 loading, 2 washing, 11 elution vessels. SX - 4 extraction, 4 strip mixer settlers.		
Resin /Solvent	Conventional strong base resin. Tertiary amine solvent.		
Elution /Stripping	10% sulphuric acid elution. Controlled pH ammonia strip.		
Precipitation Type	Conventional precipitation with ammonia gas in 3 tanks in series.		
Drying/Calcination	Hearth type roaster.		
Product	Calcined uranium oxide concentrates in drums.		
Unusual Features	Screen mix RIP. December 1978.		

103 - 15	UNION CARBIDE - PALANGANA	South Texas	U.S.A.
Owner/Operator	Union Carbide Corporation.		
Stage of Exploitation	New plant recently commissioned		
Grade (Range)	Not available.		
Production (Range)	Solution mining. Uranium production rate not available.		
Deposit Ore Type	Sandstone - roll type hosted in fluvial channel sandstones within the Whitsett Formation.		
Uraniferous Minerals	Coffinite and uraninite. Uranium also in carbonaceous associations.		
Other Minerals	Disseminated pyrite and marcasite. Molybdenum.		
Mining	Solution mining operation.		
Sorting/Blending	Not applicable.		
Preconcentration	Not applicable.		
Crushing	Not applicable.		
Grinding	Not applicable.		
Degree of Comminution	Not applicable.		
Leach Type	Solution mining using carbonate leach solution (in-situ leach)		
Oxidant/Temperature (Range)	Hydrogen peroxide. Ambient temperature.		
Leach Pulp Density (Range)	Not applicable.		
Remarks on Leaching	-		
Solids/Liquid Separation	Not applicable.		
Clarification	Not applicable.		
Solids content of preg. soln	Not applicable.		
Concentration/Purification	Modified U.S.B.M. MCIX column.		
Configuration (SX, IX)	-		
Resin /Solvent	Conventional strong-base resin.		
Elution /Stripping	2 - stage proprietary elution circuit.		
Precipitation Type	Direct uranium precipitation from eluate.		
Drying/Calcination	ADU slurry shipped to Urecon for drying/calcination.		
Product	Ammonium diuranate slurry.		
Unusual Features	Novel elution circuit. One of first U.S. plants not to undertake drying/calcination on site.		

December 1976.

103-16	UNITED NUCLEAR - HOMESTAKE	New Mexico (Grants Mineral Belt)	U.S.A.
Owner/Operator	United Nuclear Corporation + Homestake Mining Company / United Nuclear Homestake Partners.		
Stage of Exploitation	1650 t/d in 1956. Subsequent expansion and changes to flow sheet to 3000 t/d.		
Grade (Range)	0.10 - 0.15% U_3O_8 .		
Production (Range)	2000 - 3000 t/d 500 - 1000 t U/a.		
Deposit Ore Type	Sandstone and Limestone Ores - Clusters of elongated pods (trend ore) and stacked layers in sandstone lenses. Todilto limestone ores.		
Uraniferous Minerals	Coffinite, uraninite, carnotite, carbonaceous materials (kerogens)		
Other Minerals	Vanadium. Calcite in limestone ores.		
Mining	Custom treatment of limestone ores in addition to several company owned mines. In 1977 ore received from about 15 mines. Underground mining.		
Sorting/Blending	Ores blended at each mine. Ores from different mines stored separately at mill. Two separate grinding and leaching circuits		
Preconcentration	No preconcentration.		
Crushing	Impact crushers (primary only)		
Grinding	Two circuits each have single stage ball mills (carbonate solutions) in closed circuit with classifiers. Circuit for limestone ores has second stage ball mill.		
Degree of Comminution	80% minus 100 - mesh (90% minus 48 - mesh, 35% minus 200 - mesh).		
Leach Type	Two stage alkaline leach. First in autoclaves then in pachucas.		
Oxidant/Temperature (Range)	Oxygen from air. 90 - 95°C for autoclaves. 75 - 80°C for pachucas.		
Leach Pulp Density (Range)	50 - 55% solids		
Remarks on Leaching	Amenable ores - set of 5 autoclaves followed by 3 pachucas in series. Difficult ores - set of 8 autoclaves followed by 6 pachucas in series.		
Solids/Liquid Separation	3 stages of filtration in rotary drum vacuum filters.		
Clarification	Extra thickener for clarification.		
Solids content of preg. soln	<20 p.p.m.		
Concentration/Purification	Precipitation of uranium directly from alkaline leach liquors using sodium hydroxide. Precipitate roasted and water leached for vanadium removal.		
Configuration (SX, IX)	-		
Resin /Solvent	-		
Elution /Stripping	-		
Precipitation Type	Sodium diuranate is redissolved in H_2SO_4 and uranium is reprecipitated using ammonia gas.		
Drying/Calcination	Multiple hearth.		
Product	Dried yellowcake in drums (only ammonium diuranate).		
Unusual Features	Unique combination of autoclaves and pachucas for leaching.		

December 1976.

103-17	U. S. STEEL - CLAY WEST	South Texas	U.S.A.
Owner/Operator	U.S. Steel + Niagara Mohawk / U.S. Steel		
Stage of Exploitation	250 000 lb U_3O_8 p.a. in 1975 being expanded to 1 million lb U_3O_8 p.a. by 1978.		
Grade (Range)	Not applicable		
Production (Range)	100 - 500 t U/a.		
Deposit Ore Type	Sandstone - Miocene Oakville sandstone - about 350ft of interbedded sands, silts and bentonite clays		
Uraniferous Minerals	Uraninite and coffinite.		
Other Minerals	Bentonitic clays.		
Mining	Solution mining.		
Sorting/Blending	Not applicable.		
Preconcentration	Not applicable.		
Crushing	Not applicable.		
Grinding	Not applicable.		
Degree of Comminution	Not applicable.		
Leach Type	Ammonium bicarbonate solution mining (in-situ leach).		
Oxidant/Temperature (Range)	Oxygen from O_2 plant. Ambient temperature.		
Leach Pulp Density (Range)	Not applicable.		
Remarks on Leaching	Reagent used for leaching are CO_2 , NH_3 , water and O_2 . Grid of wells spaced in 50ft intervals. Injection of leach solution at each grid intersection. Extraction of leach liquors at centre of each grid.		
Solids/Liquid Separation	Not applicable		
Clarification	Downflow columns containing activated carbon.		
Solids content of preg. soln	<20 p.p.m.		
Concentration/Purification	Fixed bed ion exchange.		
Configuration (SX, IX)	Conventional.		
Resin /Solvent	Strong base resin.		
Elution /Stripping	Chloride elution.		
Precipitation Type	Conventional precipitation with NH_3 .		
Drying/Calcination	Conventional.		
Product	Dried yellowcake in drums.		
Unusual Features	December 1975.		

103-18	U. S. STEEL - BURNS RANCH	South Texas	U.S.A.
Owner/Operator	U.S. Steel.		
Stage of Exploitation	Production.		
Grade (Range)	Not applicable		
Production (Range)	100 - 300 t U/a.		
Deposit Ore Type	Sandstone - Miocene Oakville sandstone.		
Uraniferous Minerals	Uraninite and coffinite.		
Other Minerals	Bentonitic clays.		
Mining	Solution mining.		
Sorting/Blending	Not applicable.		
Preconcentration	Not applicable.		
Crushing	Not applicable.		
Grinding	Not applicable.		
Degree of Comminution	Not applicable.		
Leach Type	Alkaline carbonate solution mining (in situ leach).		
Oxidant/Temperature (Range)	Oxygen. Ambient temperature.		
Leach Pulp Density (Range)	Not applicable.		
Remarks on Leaching	-		
Solids/Liquid Separation	Not applicable.		
Clarification	Carbon filters.		
Solids content of preg. soln	-		
Concentration/Purification	Porter CIX system.		
Configuration (SX, IX)	-		
Resin /Solvent	Strong base resin.		
Elution /Stripping	Chloride and sodium carbonate elution.		
Precipitation Type	Precipitation of uranium from eluate, after acidification, using MgO .		
Drying/Calcination	-		
Product	Dried yellowcake (magnesium diuranate).		
Unusual Features	December 1976		

103-19	WESTERN NUCLEAR - JEFFREY CITY	Wyoming (Gas Hills - Crooks Gap)	U.S.A.
Owner/Operator	Phelps Dodge Corporation / Western Nuclear Corporation.		
Stage of Exploitation	1957 startup at 850 t/d. Mill revamped 1974/75 to 1500 t/d. Aim by 1979 is 2 million lb U_3O_8 p.a.		
Grade (Range)	0.05 - 0.20% U_3O_8 .		
Production (Range)	1000 - 2000 t/d 100 - 500 t U/a.		
Deposit Ore Type	Sandstone - Host rock for the irregular, stratiform and small roll-type deposits is a sequence of friable, arkosic sandstones in the Battle Spring Formation.		
Uraniferous Minerals	Mineralized zones controlled by structure and presence of carbonaceous material. Uraninite Coffinite.		
Other Minerals	Pyrite, calcite, selenium and jordanite.		
Mining	In 1977 mill received ore from about 5 mines Open pit and underground.		
Sorting/Blending	No sorting. No blending.		
Preconcentration	No preconcentration.		
Crushing	No crushing equipment.		
Grinding	Ore fed directly to semi-autogenous cascade mill.		
Degree of Comminution	80% minus 35 - mesh (50% minus 65 - 100 mesh).		
Leach Type	Acid leach in mechanical agitators.		
Oxidant/Temperature (Range)	Sodium chlorate 35 - 45°C.		
Leach Pulp Density (Range)	45 - 50%.		
Remarks on Leaching			
Solids/Liquid Separation	Multistage cyclones and drag classifiers.		
Clarification	No clarification.		
Solids content of preg. soln	12 - 15% solids in pulp to RIP.		
Concentration/Purification	Screen mix RIP. Elumex configuration (RIP + SX).		
Configuration (SX, IX)			
Resin /Solvent	RIP grade strong base resin. Tertiary amine solvent.		
Elution /Stripping	10% H_2SO_4 for elution. Controlled pH ammonia strip.		
Precipitation Type	Conventional NH_3 gas.		
Drying/Calcination	6 hearth roaster.		
Product	Dried yellowcake in drums.		
Unusual Features	Screen mix RIP. December 1978.		

103-20	RANCHERS - NATURITA	Colorado (Uranium Mineral Belt)	U.S.A.
Owner/Operator	Ranchers Exploration and Development / Durita Development Corporation.		
Stage of Exploitation	Retreatment of old tailings (short term)		
Grade (Range)	0.04 - 0.06% U_3O_8 .		
Production (Range)	< 100 t U/a.		
Deposit Ore Type	Retreatment of old tailings derived from salt roasting process for Uravan sandstone ores.		
Uraniferous Minerals	Salt roast process previously converted uranium minerals to oxidized form.		
Other Minerals	Vanadium present.		
Mining	Reclamation by front end loader.		
Sorting/Blending	Not applicable.		
Preconcentration	Not applicable.		
Crushing	Not applicable.		
Grinding	Not applicable.		
Degree of Comminution	Previously ground to minus 35-mesh.		
Leach Type	Strong acid cure followed by vat leaching.		
Oxidant/Temperature (Range)	No oxidant required. Ambient temperature.		
Leach Pulp Density (Range)	Not applicable.		
Remarks on Leaching	Reclaimed tailings are agglomerated with strong sulphuric acid and placed in earthvats for subsequent counter-current percolation leaching (3 stages of washing).		
Solids/Liquid Separation	None.		
Clarification	Clarification incidental in filter-press separation of activated carbon used to remove organics from pregnant solution.		
Solids content of preg. soln	< 20 p.p.m.		
Concentration/Purification	Solvent extraction for uranium (USX) and for vanadium (VSX).		
Configuration (SX, IX)	USX - 3 extraction, 1 scrub, 3 strip. VSX - 4 extraction, 1 scrub, 2 strip.		
Resin /Solvent	Tertiary amine solvent.		
Elution /Stripping	Sodium chloride.		
Precipitation Type	Conventional precipitation of uranium with NH_3 .		
Drying/Calcination	Not applicable.		
Product	ADU slurry at 42% water in road tankers to refinery where nitric acid is added to tanker for dissolution.		
Unusual Features	Plant largely skid mounted and will be moved to Durango when the Naturita tailings have been fully treated. Novel leach process specifically designed for retreatment of tailings. December 1978.		

201-1	CNEA - SIERRA PINTADA	Mendoza (Sierra Pintada - Malargue)	ARGENTINA
Owner/Operator	Comision Nacional de Energia Atomica (CNEA) - State / Nuclear Mendoza		
Stage of Exploitation	Small plant in operation. Larger mill under construction.		
Grade (Range)	0.10 - 0.15% U_3O_8		
Production (Range)	1000 - 2000 t/d. 500 - 1000 t U/a.		
Deposit Ore Type	Sandstone - asphaltic.		
Uraniferous Minerals	Uraninite, thucolite, carnotite, tyuyamunite, uranophane, autunite.		
Other Minerals	Quartz, mica, clay, pyrite, base metal sulphides.		
Mining	Underground		
Sorting/Blending	No sorting		
Preconcentration	No preconcentration.		
Crushing	Jaw crusher for primary, cone crusher for secondary.		
Grinding	Single stage closed-circuit rod mill.		
Degree of Comminution	All minus 700 microns		
Leach Type	Acid leach in mechanical agitators.		
Oxidant/Temperature (Range)	Manganese dioxide (40% added to crusher). 45 - 55°C.		
Leach Pulp Density (Range)	50 - 55% solids.		
Remarks on Leaching	Relatively high carbonate in ore and presence of thucolite means high acid consumption.		
Solids/Liquid Separation	1 stage thickener. 2 stages filtration.		
Clarification	Gravity sand clarifiers.		
Solids content of preg. soln	< 20 p.p.m.		
Concentration/Purification	Solvent extraction in pump-mixer settlers.		
Configuration (SX, IX)	4 extraction, 3 stripping.		
Resin /Solvent	Tertiary amine solvent.		
Elution /Stripping	Carbonate strip.		
Precipitation Type	Eluate is acidified before uranium precipitation with NaOH in steam heated vessels.		
Drying/Calcination	Radiant electric heaters.		
Product	Dried yellowcake (sodium diuranate) in drums.		
Unusual Features			

April 1978.

203-1	NUCLEBRAS	POCOS DE CALDAS	Minas Gerais	BRAZIL
Owner/Operator	Empresas Nucleares Brasileiras S.A. (Nuclebras)			
Stage of Exploitation	New mill under construction. Scheduled start up in 1979.			
Grade (Range)	0.05 - 0.10% U_3O_8			
Production (Range)	1000 - 2000 t/d. 100 - 500 t U/a.			
Deposit Ore Type	Sandstone			
Uraniferous Minerals	Coffinite			
Other Minerals				
Mining	Open pit.			
Sorting/Blending				
Preconcentration				
Crushing				
Grinding				
Degree of Comminution				
Leach Type				
Oxidant/Temperature (Range)				
Leach Pulp Density (Range)				
Remarks on Leaching				
Solids/Liquid Separation				
Clarification				
Solids content of preg. soln				
Concentration/Purification				
Configuration (SX, IX)				
Resin /Solvent				
Elution /Stripping				
Precipitation Type				
Drying/Calcination				
Product				
Unusual Features	Full details on mineralogy and metallurgical process to be used not published yet.			

April 1978

3 0 3 - 1	ALUSUISSE - BAKOUNA	Mbomou (Bakouma)	CENTRAL AFRICAN EMPIRE
Owner/Operator	Alusuisse + Cogema + the State / Societe d'Uranium Centrafricaine (URCA).		
Stage of Exploitation	Feasibility study favourable. Scheduled production 1980/81.		
Grade (Range)	0.20 - 0.30% U_3O_8 .		
Production (Range)	1000 - 1500 t U/a during first 5 year period. Thereafter 700t U/a.		
Deposit Ore Type	Phosphates - The Bakouma basin uranium is associated with phosphate lenses interstratified with the silts and siliceous horizons of the M'Patou formation. Average depth 35 - 40m. Several small deposits - Patricia, Patou, Pema, Palmyre, Ramaia and Paquerette.		
Uraniferous Minerals	Fluorapatite, (tetravalent uranium), autunite, torbenite.		
Other Minerals	Apatite, chert, clay, crandallite.		
Mining			
Sorting/Blending			
Preconcentration			
Crushing			
Grinding			
Degree of Comminution			
Leach Type			
Oxidant/Temperature (Range)			
Leach Pulp Density (Range)			
Remarks on Leaching			
Solids/Liquid Separation			
Clarification			
Solids content of preg. soln			
Concentration/Purification			
Configuration (SX, IX)			
Resin /Solvent			
Elution /Stripping			
Precipitation Type			
Drying/Calcination			
Product			
Unusual Features	The Bakouma deposits are unusual, if not unique, in character. Details of metallurgical response of ore not published yet.		

April 1978

3 0 5 - 1	CORUF - MOUNANA	Haut - Congo (Franceville)	GABON
Owner/Operator	Compagnie des mines d'uranium de Franceville (CORUF)		
Stage of Exploitation	Crude calcium uranate produced since 1981. Plant expansion and SX by 1978.		
Grade (Range)	0.30 - 0.50% U_3O_8 .		
Production (Range)	850 - 950 t/d. 1000 t U/a.		
Deposit Ore Type	Sandstone - Uranium occurs in the upper part of the basal Francevillian sandstone. Host rocks conglomerates and coarse sandstones cemented by asphaltic organic matter.		
Uraniferous Minerals	In the oxidized zone francevillite, vanuranilite and other hydrated vanadates of uranium, barite and aluminium are found. Below the oxidized zone pitchblende with accessory coffinite occurs.		
Other Minerals	Pyrite, marcasite, melnicovite, galena, sphalerite, chalcocite, and chalcocite. Dominant gangue minerals are calcite and barite. Various vanadium minerals notably oxides.		
Mining	Open pit and underground.		
Sorting/Blending	No sorting. No blending.		
Preconcentration	No preconcentration.		
Crushing	Jaw crusher for primary, gyratory crushers for secondary and tertiary.		
Grinding	Two stage (rod and ball mills) in closed circuit with classification.		
Degree of Comminution	80% minus 380 microns.		
Leach Type	Acid leach.		
Oxidant/Temperature (Range)	Sodium chlorate. 35°C.		
Leach Pulp Density (Range)	53% solids.		
Remarks on Leaching			
Solids/Liquid Separation	Single stage classifier to effect prior separation at 150 microns, followed by 4 stages of rake classifiers for sandwashing and 5 stage CCD in conventional thickeners for fines washing.		
Clarification	Downflow columns with anthracite coal.		
Solids content of preg. soln	<20 p.p.m.		
Concentration/Purification	Solvent extraction in Krebs Mixer-settlers.		
Configuration (SX, IX)	4 stage extraction, 4 stage stripping.		
Resin /Solvent	Tertiary amine solvent.		
Elution /Stripping	Sodium chloride stripping.		
Precipitation Type	Precipitation with magnesia in stirred tanks.		
Drying/Calcination	Hot air furnace.		
Product	Dried yellowcake (magnesium diuranate) in drums.		
Unusual Features	Equatorial climate - high temperatures and relative humidity.		

April 1

3 0 8 - 1	OMNIS - FORT DAUPHIN	Fort Dauphin	MALAGASY REPUBLIC
Owner/Operator	Office Militaire National pour les Industries Strategiques (OMNIS)		
Stage of Exploitation	Intermittantly exploited by CEA in 1950's and 1960's. No production at present.		
Grade (Range)	Variable grades of uranium and thorium.		
Production (Range)	During 14 years production 5900 t concentrates produced containing 1030 t U and 3200 t Th.		
Deposit Ore Type	Pagmatite - Fort Dauphin uranothorianite field consists of numerous lenticular zones of radioactive mineralization within larger lenses and bands of pyroxenite over an area of 80 by 30 km		
Uraniferous Minerals	Uranothorianite.		
Other Minerals	Pyroxenes, anorthite, mica, calcite.		
Mining	No mining at present. Uranium activities resumed with OMNIS, set up in 1975. Exploration programme underway.		
Sorting/Blending			
Preconcentration			
Crushing			
Grinding			
Degree of Comminution			
Leach Type			
Oxidant/Temperature (Range)			
Leach Pulp Density (Range)			
Remarks on Leaching			
Solids/Liquid Separation			
Clarification			
Solids content of preg. soln			
Concentration/Purification			
Configuration (SX, IX)			
Resin /Solvent			
Elution /Stripping			
Precipitation Type			
Drying/Calcination			
Product			
Unusual Features			

April 1979

3 1 1 - 1	RIO TINTO - ROSSING	Roosburg	SOUTH WEST AFRICA (NAMIBIA)
Owner/Operator	Rio Tinto Zinc + Rio Algom + Total + small shareholders/ Rossing Uranium Limited.		
Stage of Exploitation	Commissioned 1976/77. Modifications to plant in progress to improve efficiency.		
Grade (Range)	Wide range of grades exist in deposit from < 0.05 to $0.20\% \text{ U}_3\text{O}_8$.		
Production (Range)	40 000 - 50 000 t/d. 4000 - 5000 t U/a.		
Deposit Ore Type	Granite - Situated along the northern limb of a complex synclinalorium in the central portion of the Damara orogenic belt. Bulk of uranium mineralization is in alaskite that is preferentially emplaced in the heavily folded gneisses, schists, marbles and limestones of the Khan and Rossing Formations.		
Uraniferous Minerals	Primary uranium minerals uraninite and betafite (minor) give rise to secondary uranium minerals that occur along cracks and as thin films or occasional discrete crystals. Beta-uranophane and minor meta-autonite, metatorbernite and francolite are the secondary minerals.		
Other Minerals	Alaskite is a granitic rock composed almost entirely of quartz and potassium feldspar. Zircon, calcite and apatite and minor monazite, pyrite, chalcocite, bornite, molybdenite, arsenopyrite and fluorite occur. Betafite contains niobium, titanium, tantalum and tungsten.		
Mining	Open pit. Very large haulage trucks.		
Sorting/Blending	Because of the variable nature of the ore, investigations are underway to introduce the use of scintillometers for monitoring trucks as a means of rejecting waste that is unavoidably introduced during mining.		
Preconcentration	No preconcentration.		
Crushing	Conventional primary, secondary and tertiary crushing.		
Grinding	Single stage open circuit rod mills.		
Degree of Comminution	Coarse grind to minus 6-mesh. Finer grind under consideration not to increase extraction but to improve ease of handling of milled slurry.		
Leach Type	Acid leach in mechanical agitators (air lances at option of operator)		
Oxidant/Temperature (Range)	Manganese dioxide. 45 - 50°C range.		
Leach Pulp Density (Range)	> 75% solids		
Remarks on Leaching	The uraninite is accessible to leaching at a much coarser grind than the expected liberation size because of mineralogy of ore (cracks). pH value is closely controlled using electrodeless conductivity probes. Calcite content in feed monitored and acid addition adjusted accordingly to ensure optimum acid consumption at all times.		
Solids/Liquid Separation	Sand/slime separation in hydrocyclones. Sands washed in 2 stage rotoscopes. Slimes are washed in a CCD circuit, conventional thickeners. (3 stage being converted to 5 stage).		
Clarification	No clarification.		
Solids content of preg. soln	20 - 1000 p.p.m. range		
Concentration/Purification	Porter CIX system. Eluex/Bufflex configuration (CIX - SX).		
Configuration (SX, IX)	5 absorption vessels (4 active), 1 resin trap vessel, 3 fixed-bed columns for elution.		
Resin /Solvent	Strong-base resin. Tertiary amine solvent.		
Elution /Stripping	10% H_2SO_4 elution. Controlled pH ammonia strip.		
Precipitation Type	Conventional NH_3 gas precipitation.		
Drying/Calcination	Multiple hearth (Skinner roaster)		
Product	Calcined uranium oxide concentrate in drums.		
Unusual Features	Tonnage of ore treated per day is an order of magnitude greater than that for most uranium ore processing plants. Unusual process features are coarse grind, rotoscopes and very large Porter CIX system.		

April 1

312 - 1	SOMAIR - ARLIT	Azbine (Agades)	NIGER
Owner/Operator	Niger government + COGEMA + CPMU + Mokta + Minatome / Societe des Mines de l'Air (Somaif) + Uranggesellschaft + Agip Nucleare		
Stage of Exploitation	Routine production. Start-up 1971. Expansion under consideration.		
Grade (Range)	0.20 - 0.30% U_3O_8		
Production (Range)	2000 - 3000 t/d. 1000 - 2000 t U/a.		
Deposit Ore Type	Sandstone - alternate layers of compacted, interstratified sandstones and clays. Deposit occurs in the Agades basin on the southern and western slopes of the Air mountain range.		
Uraniferous Minerals	70 - 80% of uranium in sandstones as pitchblende and coffinite. 20 - 30% of uranium in clays as organo-uranium compounds.		
Other Minerals	Pyrite, baryte, kaolinite. Calcite and vanadium very low at Arlit.		
Mining	Open pit.		
Sorting/Blending	No sorting.		
Preconcentration	No preconcentration.		
Crushing	Primary crushing only.		
Grinding	Single stage autogenous (dry) Cascade type mill.		
Degree of Comminution	80% minus 600 microns.		
Leach Type	Strong acid leaching (granule impregnation process involving curing).		
Oxidant/Temperature (Range)	No oxidant added. 75 - 80°C.		
Leach Pulp Density (Range)	Not applicable.		
Remarks on Leaching	Ground ore is preheated in a Louisville tube, then impregnated with 50% H_2SO_4 in a granulation drum for 3 hours and then ground in a pebble mill before repulping with water to wash out solubilized uranium.		
Solids/Liquid Separation	Single stage rake classifiers to effect prior separation at 150 microns		
Clarification	5-stage rake classifiers for sands washing. 6-stage CCO in thickeners for fines. Sand classifiers.		
Solids content of preg. soln	< 20 p.p.m.		
Concentration/Purification	Solvent extraction in mixer-settlers.		
Configuration (SX, IX)	4 stages extraction, 3 stages stripping.		
Resin /Solvent	Tertiary amine solvent.		
Elution /Stripping	Sodium carbonate.		
Precipitation Type	Precipitation with sodium hydroxide in stirred tanks.		
Drying/Calcination	Belt type dryer.		
Product	Dried yellowcake (sodium diuranate) in drums.		
Unusual Features	Remote location - High transport costs, special protection for equipment against solar radiation. Strong acid leach process and equipment of particular interest.		

April 1970

312 - 2	COMINAK - AKOUTA	Azbine (Agades)	NIGER
Owner/Operator	Cogema .Niger government + O.U.R.D.C. of Japan + ENUSA / Compagnie Miniere D'Akouta (Cominak)		
Stage of Exploitation	New plant scheduled for start-up 1978.		
Grade (Range)	0.40% - 0.50% U_3O_8		
Production (Range)	1000 - 2000 t/d. 2000 t U/a.		
Deposit Ore Type	Sandstone - alternate layers of compacted, interstratified sandstone and clays.		
Uraniferous Minerals	Sandstones contain pitchblende and coffinite. Clays contain organo-uranium compounds.		
Other Minerals	Pyrite, baryte, kaoline. High molybdenum content		
Mining	Underground mining.		
Sorting/Blending	No sorting.		
Preconcentration	No preconcentration.		
Crushing	Primary crushing only using jaw crusher.		
Grinding	Single stage semi-autogenous (dry) mill.		
Degree of Comminution	95% minus 600 microns.		
Leach Type	Strong acid leach (granule impregnation process involving curing).		
Oxidant/Temperature (Range)	Sodium chlorate. 80°C.		
Leach Pulp Density (Range)	Not applicable.		
Remarks on Leaching	Ore is impregnated in a granulation drum, then cured on a horizontal conveyor belt for 3 hours. Otherwise as for Arlit.		
Solids/Liquid Separation	Filtration using horizontal belt filters.		
Clarification	Downflow columns with anthracite coal.		
Solids content of preg. soln	< 20 p.p.m.		
Concentration/Purification	Solvent extraction in mixer-settlers.		
Configuration (SX, IX)	4 stage extraction, 4 stage stripping.		
Resin /Solvent	Tertiary amine solvent.		
Elution /Stripping	Sodium chloride stripping for uranium. Sodium carbonate stripping for Mo.		
Precipitation Type	Precipitation with magnesia in stirred tanks.		
Drying/Calcination	Atomisation dryer (STEC patent)		
Product	Dried yellow cake (magnesium diuranate) in drums.		
Unusual Features	Same as Arlit. In addition high Mo content in feed required more complex SX plant for Mo removal		

April 1970

3 1 2 - 3	IMOURAREN	Azbine (Agades)	NIGER
Owner/Operator	Niger government + Conoco + Cogema.		
Stage of Exploitation	Ore processing investigation and planning in progress. Target date for start-up 1982/83.		
Grade (Range)	0.12 - 0.18% U_3O_8 .		
Production (Range)	8400 t/d. 3000 t U/a.		
Deposit Ore Type	Sandstone - sandstone only containing no clay.		
Uraniferous Minerals	Uranophane.		
Other Minerals	Chrysocolla, hematite, chalcocite. Natural zeolite and copper minerals.		
Mining	Open pit mining.		
Sorting/Blending	No sorting.		
Preconcentration	No preconcentration.		
Crushing	No decision yet.		
Grinding	No decision yet.		
Degree of Comminution	95% minus 600 microns.		
Leach Type	Acid leaching with high terminal free acidity (40 g/l).		
Oxidant/Temperature (Range)	Potassium chlorate. 65°C.		
Leach Pulp Density (Range)	60 - 65% solids.		
Remarks on Leaching	-		
Solids/Liquid Separation	Horizontal belt filters under consideration.		
Clarification	No decision yet.		
Solids content of preg. soln	< 20 p.p.m.		
Concentration/Purification	Solvent extraction in mixer-settlers.		
Configuration (SX, IX)	4 stage extraction. 4 stage stripping.		
Resin /Solvent	Tertiary amine solvent.		
Elution /Stripping	Sodium chloride stripping.		
Precipitation Type	Precipitation with magnesium hydroxide in stirred tanks.		
Drying/Calcination	No decision yet on type of dryer.		
Product	Dried yellowcake (magnesium diuranate).		
Unusual Features	April 1978.		

3 1 2 - 4	ARBINE	Azbine (Agades)	NIGER
Owner/Operator	Information not available.		
Stage of Exploitation	Under investigation by Cogema.		
Grade (Range)	0.30 - 0.40% U_3O_8 .		
Production (Range)	Probably 1400 - 1600 t/d and 1000 - 1500 t U/a.		
Deposit Ore Type	Sandstone.		
Uraniferous Minerals	Coffinite, Pitchblende.		
Other Minerals	Pyrite. High in molybdenum.		
Mining	Will be open pit mining.		
Sorting/Blending	No sorting.		
Preconcentration	No preconcentration.		
Crushing	Not decided yet.		
Grinding	Not decided yet.		
Degree of Comminution	Not decided yet.		
Leach Type	Strong acid leach plus conventional acid leach.		
Oxidant/Temperature (Range)	Ammonium nitrate. 90°C.		
Leach Pulp Density (Range)	60 - 65% range.		
Remarks on Leaching	15 - 30 minutes residence time for strong acid (granule impregnation) leach followed by 2 - 6 hours conventional acid leach at high terminal acidity (no curing step).		
Solids/Liquid Separation	Horizontal belt filters.		
Clarification	Not decided yet.		
Solids content of preg. soln	< 20 p.p.m.		
Concentration/Purification	Solvent extraction in mixer-settlers.		
Configuration (SX, IX)	4 stage extraction, 4 stages stripping.		
Resin /Solvent	Tertiary amine solvent.		
Elution /Stripping	Sodium chloride for U, Sodium carbonate for Mo stripping.		
Precipitation Type	Precipitation with magnesia in stirred tanks.		
Drying/Calcination	Not decided yet.		
Product	Dried yellowcake (magnesium diuranate) in drums.		
Unusual Features	Note : Above information subject to change - project still in initial stages.		

April 1978.

314-1	ANGLO - VAAL REEFS	Transvaal (Klerksdorp)	SOUTH AFRICA								
Owner/Operator	Anglo American Corporation / Vaal Reefs Exploration and Mining Company Ltd.										
Stage of Exploitation	Original Vaal Reefs uranium plant 1956. SX introduced 1970. Merged with Western Reefs 1972 West and East uranium plants extended and new South plant 1977/79										
Grade (Range)	<0.05% U_3O_8										
Production (Range)	West plant (10 000 t/d), East plant (7000 t/d), South (8000 t/d) Total Uranium production for the three plants : 1000 - 2000 t U/a										
Deposit Ore Type	Conglomerate - uranium occurs with gold in quartz-pebble conglomerates in the Upper Division of the Witwatersrand System (mainly Vaal Reef)										
Uraniferous Minerals	Uraninite, Brannerite										
Other Minerals	Gold (major), silver, quartz, pyrite										
Mining	Underground and reclaimed slimes from surface accumulations										
Sorting/Blending	Hand sorting										
Preconcentration	Heavy media separation plant operated at East Plant during the early 1970's. Flotation for pyrite (after gold and uranium extraction) at East and West Plants										
Crushing	Jaw for primary, cone crushers for secondary and tertiary										
Grinding	2 stage grind in closed circuit with cyclones using ball and pebble mills										
Degree of Comminution	<table border="1"> <thead> <tr> <th></th><th>WEST PLANT</th><th>EAST PLANT</th><th>SOUTH PLANT</th></tr> </thead> <tbody> <tr> <td>+150 7</td><td>-150+75 28</td><td>-75µm 65</td><td></td></tr> </tbody> </table>				WEST PLANT	EAST PLANT	SOUTH PLANT	+150 7	-150+75 28	-75µm 65	
	WEST PLANT	EAST PLANT	SOUTH PLANT								
+150 7	-150+75 28	-75µm 65									
Leach Type	Acid leach in air agitated pachuca	Acid leach in air agitated pachuca	Acid leach in air agitated pachuca								
Oxidant/Temperature (Range)	MnO_2 / 50°C	MnO_2 / 50°C	MnO_2 / 60°C								
Leach Pulp Density (Range)	60% solids	60% solids	60% solids								
Remarks on Leaching			Reverse Leach								
Solids/Liquid Separation	2 stage filtration in rotary drum vacuum filters	Combination of rotary drum and horizontal belt vacuum filters	6 stage CCD circuit with conv. thickeners								
Clarification	Leaf clarifiers and old ion exchange columns	Leaf clarifiers and old ion exchange columns	No clarification								
Solids content of preg. soln	<20 p.p.m.	20 p.p.m.	20 - 1000 p.p.m.								
Concentration/Purification	Solvent extraction (Purlex) in mixer settlers	Solvent extraction (Purlex) in mixer settlers	Continuous ion exchange (Himley CIX) + SX in Bufflex configuration								
Configuration (SX, IX)			CIX : Absorption column, measuring chamber, elution column, rinse chamber; SX : 3 stages extraction, 3 stages scrubbing + 3 stripping								
Resin /Solvent	Tertiary amine solvent	Tertiary amine solvent	Conv. strong-base resin, tertiary amine solvent.								
Elution /Stripping	Controlled pH ammonia strip	Controlled pH ammonia strip	10% H_2SO_4 elution, controlled pH ammonia strip								
Precipitation Type	Uranium precipitation with ammonia gas in conventional stirred tanks										
Drying/Calcination											
Product	Yellowcake slurry transported in road tankers to central calcining plant (NUFCOR)										
Unusual Features	Largest gold mine in the world in terms of tonnage treated. After commissioning of South plant, largest single uranium producer in South Africa December 1978										

314-2	ANGLO - WESTERN DEEP LEVELS	Transvaal (West Witwatersrand)	SOUTH AFRICA								
Owner/Operator	Anglo American Corporation/ Western Deep Levels Ltd.										
Stage of Exploitation	Original plant start up 1970. Expansion 2000 t/d to 9000 t/d 1979/80										
Grade (Range)	< 0.05% U_3O_8										
Production (Range)	9000 - 10000 t/d 200 - 500 t U/a (after expansion)										
Deposit Ore Type	Conglomerate - uranium occurs with gold in quartz-pebble conglomerates in the Upper Division of the Witwatersrand system (mainly Carbon Leader and Ventersdorp Contact reefs)										
Uraniferous Minerals	Uraninite, thucholite										
Other Minerals	Gold (major), silver (minor), quartz, pyrite, dolomite.										
Mining	Underground and reclaimed slimes from surface accumulations.										
Sorting/Blending	Hand sorting. Radiometric sorting under investigation.										
Preconcentration	Screening at 3mm results in concentration of 50% of total uranium in undersize fraction which represents only 30% of mass of run-of-mine ore										
Crushing	Jaw for primary. Cone crushers for secondary and tertiary.										
Grinding	2 stage grind. Ball mills in closed circuit with classifiers followed by pebble mills in closed circuit with hydrocyclones										
Degree of Comminution	<table border="1"> <thead> <tr> <th></th><th>+150 7</th><th>-150+75 28</th><th>-75µm 65</th></tr> </thead> <tbody> <tr> <td></td><td>0,3</td><td>7,1</td><td>92,6</td></tr> </tbody> </table>				+150 7	-150+75 28	-75µm 65		0,3	7,1	92,6
	+150 7	-150+75 28	-75µm 65								
	0,3	7,1	92,6								
Leach Type	Acid leach in air agitated pachuca										
Oxidant/Temperature (Range)	MnO_2 / 50°C										
Leach Pulp Density (Range)	60 - 65% solids										
Remarks on Leaching	Reverse leach										
Solids/Liquid Separation	2-stage filtration in rotary drum vacuum filters (CCD proposed for extension)										
Clarification	Leaf clarifiers. Pressure precoat filter										
Solids content of preg. soln	< 20 p.p.m.										
Concentration/Purification	Solvent extraction (Purlex) in mixer settlers										
Configuration (SX, IX)	4 stages extraction, 4 stages scrubbing, 4 stages stripping.										
Resin /Solvent	Tertiary amine solvent										
Elution /Stripping	Controlled pH ammonia strip with ammonium sulphate recycle										
Precipitation Type	Uranium precipitated with NH_3 gas in conventional stirred tanks										
Drying/Calcination											
Product	Yellowcake slurry transported in road tankers to central calcining plant (NUFCOR)										
Unusual Features											

314-3	ANGLO - ERGO	Transvaal (East Rand Basin)	SOUTH AFRICA
Owner/Operator	Anglo American Corp/ East Rand Gold and Uranium Company Ltd. (Ergo)		
Stage of Exploitation	Production. Plant start-up 1978.		
Grade (Range)	$< 0.05\% \text{U}_3\text{O}_8$		
Production (Range)	Feed to flotation 48 000 t/d. Pyrite concentrate 1500 t/d processed through the uranium plant. Uranium production 100 - 200 t U/a		
Deposit Ore Type	Conglomerate - The slimes that are being reclaimed resulted from past treatment (for gold) of quartz-pebble conglomerate ores from the Upper Division of the Witwatersrand system.		
Uraniferous Minerals	Uraninite		
Other Minerals	Gold (major), silver, pyrite, quartz		
Mining	Entire feed to plant is reclaimed slimes from surface accumulations, 18 old slimes dams (300 million tons) to be processed over 20 year period at the rate of 1.5 million tons per month. Water jet monitoring of slimes		
Sorting/Blending	No sorting		
Preconcentration	Flotation for production of a concentrate containing gold uranium and pyrite		
Crushing	No crushing		
Grinding	No milling		
Degree of Comminution	+150 -150+75 -75µm 5.0 25.0 95.0		
Leach Type	Acid leach of pyrite concentrates in air-agitated pachuca		
Oxidant/Temperature (Range)	No oxidant/ 55 - 60°C		
Leach Pulp Density (Range)	60% solids		
Remarks on Leaching			
Solids/Liquid Separation	2 stage filtration in rotary drum vacuum filters		
Clarification	Single stage upflow sand clarifier		
Solids content of preg. soln	<20 p.p.m.		
Concentration/Purification	Solvent extraction (Purlex) in mixer settlers		
Configuration (SX, IX)	3 stages extraction, 4 scrubbing and 4 stripping		
Resin /Solvent	Tertiary amine solvent		
Elution /Stripping	Controlled pH ammonia strip with ammonium sulphate recycle		
Precipitation Type	Uranium precipitation with NH_3 gas in conventional stirred tanks		
Drying/Calcination			
Product	Yellowcake slurry transported in road tankers to central calcining plant (NUFCOR)		
Unusual Features	Large scale reclamation of residue dams and preconcentration by flotation.		

December 1978.

December 1978.

314-4	ANGLO - PRESIDENT BRAND (Joint Metallurgical Scheme)	Orange Free State (Free State Goldfields)	SOUTH AFRICA						
Owner/Operator	Anglo American Corp./President Brand G.M. Co. (Joint Metallurgical Scheme)								
Stage of Exploitation	Uranium plant erected at President Brand in 1971 not commissioned immediately. Plant expanded and commissioned 1977/78 as focal point for Joint Metallurgical Scheme.								
Grade (Range)	< 0.05% U_3O_8								
Production (Range)	Combined uranium produced from President Brand, Free State Geduld, Western Holdings, President Steyn and Welkom Mines : 500 - 1000 t U/a.								
Deposit Ore Type	Conglomerate - uranium occurs with gold in quartz-pebble conglomerates in the Upper Division of the Witwatersrand System (mainly Basal, Leader and Elsburg Reefs)								
Uraniferous Minerals	Uraninite, Brannerite, Thucolite.								
Other Minerals	Gold (major), silver, quartz, pyrite, pyrophyllite								
Mining	Mostly reclaimed slimes from surface accumulations but some current residues from gold plants treated for uranium. Underground mining.								
Sorting/Blending	Hand sorting at the various gold mines that provide feed to the Joint Metallurgical Scheme.								
Preconcentration	Flotation at pH 4 with sodium mercaptobenzoethiazole as collector, and IEB as frother is used to produce pyrite concentrates that are acid leached for uranium recovery.								
Crushing	Jaw for primary, cone crushers for secondary and tertiary								
Grinding	Two stage grind rod, ball and pebble mills in closed circuit with hydrocyclones. Also run-of-mill milling								
Degree of Comminution	<table><tr><td>+150</td><td>-150+75</td><td>-75µm</td></tr><tr><td>8.1</td><td>22.3</td><td>71.5</td></tr></table>			+150	-150+75	-75µm	8.1	22.3	71.5
+150	-150+75	-75µm							
8.1	22.3	71.5							
Leach Type	Acid leaching in air agitated pachuca								
Oxidant/Temperature (Range)	H_2O_2 / 60 - 65°C								
Leach Pulp Density (Range)									
Remarks on Leaching									
Solids/Liquid Separation	Two stage filtration in rotary drum vacuum filters								
Clarification	Gravity sedimentation followed by pressure precoat filters								
Solids content of preg. soln	< 20 p.p.m.								
Concentration/Purification	Solvent extraction (Purlex) in mixer-settlers								
Configuration (SX, IX)	4 stages extraction, 4 stages scrubbing, 4 stages stripping								
Resin /Solvent	Tertiary amine solvent								
Elution /Stripping	Controlled pH ammonia strip with ammonium sulphate recycle								
Precipitation Type	Uranium precipitation from strip solution with NH_3 gas in conventional stirred tanks								
Drying/Calcination									
Product	Yellowcake slurry transported in road tankers to central calcining plant (NUFCOR)								
Unusual Features	Metallurgical accounting system has been designed to distribute costs and revenues from the complex as a whole among the various gold mines that contribute feed material to the Scheme								

December 1978

December 1978

314-5	ANGLOVAAL - HAARTEBEEST - FONTEIN	Transvaal (Klerksdorp)	SOUTH AFRICA
Owner/Operator	Anglo-Transvaal Consolidated Investment /Haartebeestfontein G.M. Co. Ltd.		
Stage of Exploitation	Original uranium plant start-up 1955. Expansion 8500 t/d to 10 000 t/d 1978/79		
Grade (Range)	< 0.05% U_3O_8		
Production (Range)	9000 - 10 000 t/d 200 - 500 t U/a.		
Deposit Ore Type	Conglomerate - uranium occurs with gold in quartz pebble conglomerates in the Upper Division of the Witwatersrand system. (Mainly Vaal Reef).		
Uraniferous Minerals	Uraninite		
Other Minerals	Gold (major) silver (minor) quartz, pyrite.		
Mining	Underground and reclaimed slimes from surface accumulations.		
Sorting/Blending	Hand sorting (reef picking)		
Preconcentration	Flotation for pyrite concentrate which is reground and acid leached for uranium and cyanide-leached for gold prior to sale as by-product.		
Crushing	Jaw crushers for primary, cone crushers for secondary and tertiary.		
Grinding	Three stage grind in closed circuit with classification using rod and pebble mills.		
Degree of Comminution	60% minus 200 mesh		
Leach Type	Acid leach in air agitated pachuca		
Oxidant/Temperature (Range)	Ferric iron + oxygen from air / 55 - 65°C		
Leach Pulp Density (Range)	55 - 80% solids		
Remarks on Leaching	Ferric leach for uranium		
Solids/Liquid Separation	3 - stage filtration in rotary drum vacuum filters		
Clarification	Sand clarifiers		
Solids content of preg. soln	< 50 p.p.m.		
Concentration/Purification	Fixed bed ion-exchange		
Configuration (SX, IX)	2 stages absorption and 1 stage elution		
Resin /Solvent	Conventional strong-base resin.		
Elution /Stripping	Ammonium nitrate elution		
Precipitation Type	Fe precipitation with CaO before uranium precipitation with NH_3 gas in conventional stirred tanks		
Drying/Calcination			
Product	Yellowcake slurry transported in road tankers to central calcining plant (NUFCOR)		
Unusual Features	Autoxidation process using SO_2 for regeneration of ferric iron from ferrous iron in recycle solution.		

314-6	GENERAL MINING - BUFFELS - FONTEIN	Transvaal (Klerksdorp)	SOUTH AFRICA
Owner/Operator	General Mining and Finance Corporation/ Buffelsfontein G.M. Co. Ltd.		
Stage of Exploitation	Production. Original uranium plant start-up 1957. Expansion and change to Purlex 1968.		
Grade (Range)	< 0.05% U_3O_8		
Production (Range)	8000 - 9000 t/d 500 - 1000 t U/a		
Deposit Ore Type	Conglomerate - uranium occurs with gold in quartz-pebble conglomerates in the Upper Division of the Witwatersrand system. (Mainly Vaal Reef)		
Uraniferous Minerals	Uraninite		
Other Minerals	Gold (major), silver (minor), quartz, pyrite, dolomite		
Mining	Underground and reclaimed slimes from surface accumulations		
Sorting/Blending	Hand sorting (waste picking) + photometric sorting + Reef picking on waste sorted		
Preconcentration	None		
Crushing	Jaw for primary, cone crushers for secondary and tertiary		
Grinding	Open circuit rod mills followed by pebble mills in closed circuit with hydrocyclones		
Degree of Comminution	70% minus 200 mesh		
Leach Type	Acid leach in air agitated pachuca		
Oxidant/Temperature (Range)	MnO_2 / 65°C also ferric iron		
Leach Pulp Density (Range)	60 - 65% solids		
Remarks on Leaching	Oxidants and acid added to first two pachuca for ferric leach/continuous flow		
Solids/Liquid Separation	2-stage filtration in rotary drum vacuum filters + 2 x 70m ² horizontal belt filters.		
Clarification	Sand clarifiers + upflow sand filters.		
Solids content of preg. soln	20 ppm to 80 ppm.		
Concentration/Purification	Solvent extraction (Purlex) in mixer settlers		
Configuration (SX, IX)	4 stages extraction, 5 stages scrubbing, 4 stages stripping.		
Resin /Solvent	Tertiary amine solvent		
Elution /Stripping	Controlled pH ammonia strip with ammonium sulphate recycle		
Precipitation Type	Uranium precipitation with NH_3 gas in conventional stirred tanks		
Drying/Calcination			
Product	Yellowcake slurry transported in road tankers to central calcining plant (NUFCOR)		
Unusual Features	Reverse leach process incorporating recovery of uranium then gold then flotation of pyrite for sulphuric acid production. Bacterial oxidation of Fe^{2+} to Fe^{3+} in solutions recycled to leach		

December 1976

314 - 7	GENERAL MINING - W. R. CONSOLIDATED DATED	Transvaal (Central and Near West Rand)	SOUTH AFRICA
Owner/Operator	General Mining and Finance Corporation/ West Rand Consolidated Mines Ltd.		
Stage of Exploitation	Production. Original uranium plant 1952. Change from IX to SX 1970. Production interrupted 1974 - 76.		
Grade (Range)	< 0.05% U_3O_8		
Production (Range)	2000 - 3000 t/d 200 - 500 t U/a		
Deposit Ore Type	Conglomerate - Uranium occurs in quartz pebble conglomerates in the Upper Division of the Witwatersrand system. (Mainly White, Monarch, and one or two other reefs of the Bird Reef Group)		
Uraniferous Minerals	Uraninite		
Other Minerals	Gold (minor) silver (minor), quartz, pyrite		
Mining	Underground.		
Sorting/Blending	Hand sorting and radiometric sorting, waste picking.		
Preconcentration	None		
Crushing	Jaw for primary, cone crushers for secondary and tertiary.		
Grinding	2 stage grind in closed circuit with classifiers. Ball and pebble mills		
Degree of Comminution	60 - 65% minus 200 mesh		
Leach Type	Acid leach in air-agitated pachucas		
Oxidant/Temperature (Range)	HNO_3 / 55°C		
Leach Pulp Density (Range)	60 - 65% solids		
Remarks on Leaching	Batch operation		
Solids/Liquid Separation	2-stage filtration in rotary drum vacuum filters		
Clarification	Sand clarifiers		
Solids content of preg. soln	< 20 p.p.m.		
Concentration/Purification	Solvent extraction (Purlex) in mixer settlers		
Configuration (SX, IX)			
Resin /Solvent	Tertiary amine solvent		
Elution /Stripping	Controlled pH ammonia strip with ammonium sulphate recycle		
Precipitation Type	Uranium precipitated with NH_3 gas in conventional stirred vessels		
Drying/Calcination			
Product	Yellowcake slurry transported in road tankers to central calcining plant (NUFCOR)		
Unusual Features	The first uranium extraction plant to be constructed in South Africa and also the first to justify mining a particular reef series for its uranium (rather than its gold) content. December 1976		

314 - 8	GOLD FIELDS - WEST ORE - FONTEIN	Transvaal (West Witwatersrand)	SOUTH AFRICA
Owner/Operator	Gold Fields of South Africa Ltd/ West Orlfontein G.M. Co. Ltd.		
Stage of Exploitation	Production. Original plant 1956-62. Present plant started up 1971.		
Grade (Range)	* 0.05% U_3O_8		
Production (Range)	2000 - 3000 t/d 200 - 500 t U/a.		
Deposit Ore Type	Conglomerate - uranium occurs with gold in quartz-pebble conglomerates in the Upper Division of the Witwatersrand system. (Mainly Caroon Leader and Ventersdorp contact reefs)		
Uraniferous Minerals	Uraninite		
Other Minerals	Gold (major) silver (minor) quartz, pyrite.		
Mining	Underground and reclaimed slimes from surface accumulations		
Sorting/Blending	Hand sorting (reef picking). Photometric sorting		
Preconcentration	Screening at 3mm results in concentration of 60% of total uranium in undersize fraction which represents only 30% of mass of run-of-mine ore.		
Crushing	Jaw for primary, and cone crushers for secondary and tertiary.		
Grinding	Two stage grind in closed circuit with classification using rod, ball and pebble mills.		
Degree of Comminution	80% minus 200 mesh		
Leach Type	Acid leach in air-agitated pachucas		
Oxidant/Temperature (Range)	HNO_3 / 50 - 60°C		
Leach Pulp Density (Range)	60 - 65% solids		
Remarks on Leaching			
Solids/Liquid Separation	2 stage filtration in rotary drum vacuum filters		
Clarification	Pressure precoat filter (Fundal). Diatomaceous earth precoat		
Solids content of preg. soln	< 10 p.p.m.		
Concentration/Purification	Solvent extraction (Purlex) in mixer settlers		
Configuration (SX, IX)	4 stages absorption, 5 stages scrubbing, 4 stages stripping		
Resin /Solvent	Tertiary amine solvent		
Elution /Stripping	Controlled pH ammonia strip with ammonium sulphate recycle		
Precipitation Type	Uranium precipitation with ammonia gas in conventional stirred tanks		
Drying/Calcination			
Product	Yellowcake slurry transported in road tankers to central calcining plant (NUFCOR)		
Unusual Features	December 1976.		

314 - 9	JCI - RANDFONTEIN	Transvaal (Central and near West Rand)	SOUTH AFRICA
Owner/Operator	Johannesburg Consolidated Investment Co. Ltd/ Randfontein Estates G.M. Co. Witwatersrand Ltd.		
Stage of Exploitation	Production. Two uranium plants (Millsite and Cooke) commissioned 1977/78		
Grade (Range)	< 0.05% U_3O_8		
Production (Range)	Millsite (2000 - 3000 t/d), Cooke (7000 - 8000 t/d). Total uranium production for both plants 500 - 1000 t U/a.		
Deposit Ore Type	Conglomerate - uranium occurs with gold in quartz-pebble conglomerates in the Upper Division of the Witwatersrand System (Bird and Main Reef Series)		
Uraniferous Minerals	Uraninite, brannerite, thucolite.		
Other Minerals	Gold, silver (minor), quartz, pyrite, leucocene, goethite-limonite, chlorite, sericite.		
Mining	Underground		
Sorting/Blending			
Preconcentration	No preconcentration		
Crushing	Jaw for primary, cone crushers for secondary No crushing		
Grinding	Two stage in closed circuit with classification Single stage semi-autogenous milling in closed circuit with classification.		
Degree of Comminution			
Leach Type	Acid leach in air agitated pachuca.		
Oxidant/Temperature (Range)	PnO_2 / 45 - 55°C		
Leach Pulp Density (Range)	55 - 60% solids		
Remarks on Leaching			
Solids/Liquid Separation	Single stage filtration in horizontal belt vacuum filters		
Clarification	Partial clarification in lamella thickeners.		
Solids content of preg. soln	20 - 1000 p.p.m.		
Concentration/Purification	Continuous ion exchange (Davy Powergas design) and solvent extraction in Eluex/Buflex configuration.		
Configuration (SX, IX)	Upflow CIX for absorption, fixed bed elution. SX: 4 stages extraction, 3 stages scrubbing, 4 stages stripping.		
Resin /Solvent	Conventional strong-base resin for CIX. Tertiary amine solvent for SX		
Elution /Stripping	10% H_2SO_4 elution for CIX. Controlled pH ammonia strip for SX		
Precipitation Type	Uranium precipitation with NH_3 gas in conventional stirred tanks		
Drying/Calcination			
Product	Yellowcake slurry transported in road tankers to central calcining plant (NUFCOR)		
Unusual Features	December 1978.		

314 - 10	RAND MINES - BLYVOORUIT - ZICHT	Transvaal (West Witwatersrand)	SOUTH AFRICA
Owner/Operator	Rand Mines Ltd / Blyvooruitzicht G.M. Co Ltd.		
Stage of Exploitation	Original uranium plant 1953-64. New plant 1970. Expansion and flowsheet changes 1977		
Grade (Range)	< 0.05% U_3O_8		
Production (Range)	5000 - 6000 t/d. 200 - 500 t U/a		
Deposit Ore Type	Conglomerate - uranium occurs with gold in quartz-pebble conglomerates in the Upper Division of the Witwatersrand system. (Mainly Carbon Leader and Ventersdorp contact reefs)		
Uraniferous Minerals (Major)	Uraninite, uraniferous carbon		
Other Minerals	a) Major gangue minerals : quartz, micaceous silicates b) Of economic importance : gold, silver(alloyed with gold).		
Mining	Underground and reclaimed slimes from surface accumulations		
Sorting/Blending	Hand sorting (reef picking)		
Preconcentration	Screening at 19mm results in concentration of 60% of total uranium in undersize fraction which represents only 30% of mass of run-of-mine ore.		
Crushing	Jaw crushers for primary, standard and shorthead crushers for secondary and tertiary		
Grinding	Primary rod mills and secondary pebble or ball mills each in closed circuit with hydrocyclones.		
Degree of Comminution	70% minus 200 mesh		
Leach Type	Acid leach in air-agitated pachuca		
Oxidant/Temperature (Range)	PnO_2 / ambient		
Leach Pulp Density (Range)	60% solids		
Remarks on Leaching			
Solids/Liquid Separation	Two separate circuits. First consists of two stages filtration in rotary drum vacuum filters - horizontal belt filters. Second (slimes reclamation circuit) is 5-stage CCD in conventional thickeners		
Clarification	No clarification		
Solids content of preg. soln	20 - 1000 p.p.m.		
Concentration/Purification	Continuous ion exchange (NIMEIX) and SX in Eluex/Buflex configuration.		
Configuration (SX, IX)	CIX has single stage absorption, single stage elution both continuous SX has 3 stages extraction, 5 stages scrubbing, 4 stages stripping, 1 stage solvent regeneration.		
Resin /Solvent	Conventional strong-base resin. Tertiary amine solvent.		
Elution /Stripping	10% H_2SO_4 elution for CIX. Controlled ammonia strip for SX		
Precipitation Type	Uranium precipitation with NH_3 gas in conventional stirred tanks		
Drying/Calcination			
Product	Yellowcake slurry transported in road tankers to central calcining plant (NUFCOR)		
Unusual Features	First plant in South Africa to introduce CCD and CIX. Bucketwheel excavators for slimes reclamation		
	December 1978		

314-11	RAND MINES - HARMONY	Orange Free State (Free State Goldfields)	SOUTH AFRICA
Owner/Operator	Rand Mines Ltd/ Harmony S.M. Co. Ltd.		
Stage of Exploitation	Original Harmony uranium plant start up 1955. Expansion and SX 1970. Virginia gold mine and its uranium plant (start up 1955) taken over 1973. Merriespruit gold mine taken over 1973. New uranium plant under construction at Merriespruit 1979/80.		
Grade (Range)	< 0.05% U ₃ O ₈		
Production (Range)	Harmony (7000 - 8000 t/d), Virginia (5000 - 6000 t/d), Merriespruit (7000 - 8000 t/d) Total uranium production for all 3 plants/ 500 - 1000 t U/a		
Deposit Ore Type	Conglomerate - uranium occurs with gold in quartz-pebble conglomerates in the Upper Division of the Witwatersrand system (mainly Basal, and Leader Reefs)		
Uraniferous Minerals (Major)	Uraninite		
Other Minerals	a) Major gangue minerals : Quartz, micaceous silicates b) Of economic importance : Gold, silver(alloyed with gold), pyrite.		
Mining	Underground and reclaimed slimes from surface accumulations.		
Sorting/Blending	Hand sorting (waste sorting?)		
Preconcentration	No preconcentration		
Crushing	HARMONY PLANT Jaw crushers for primary, cone crushers for secondary and tertiary.	VIRGINIA PLANT Jaw, standard and shorthead cone crushing	PROPOSED MERRIESPRUIT PLANT Jaw, standard and shorthead cone crushing
Grinding	Two stage grind in closed circuit with classifiers using ball and pebble mills	Two stage in Rod (for Ball) followed by pebble milling - Closed circuit with classifiers.	Two stage Rod and Pebble milling - closed circuit with classifiers
Degree of Comminution	70% minus 200 mesh	53% minus 200 mesh	60% minus 200 mesh
Leach Type	Acid leach in air-agitated pachucas	Acid leach (ferric) in air-agitated pachucas	Acid leach (ferric) in air-agitated pachucas
Oxidant/Temperature (Range)	MnO ₂ / ambient	60°C	60 - 65°C
Leach Pulp Density (Range)	~ 80% solids	~ 65% solids	60 - 65% solids
Remarks on Leaching			
Solids/Liquid Separation	2 stage filtration in rotary drum vacuum filters	2 stage filtration in rotary drum vacuum filters	Horizontal belt vacuum filters
Clarification	Resin beds in old fixed-bed ion exchange columns	Sand clarifiers	No clarification
Solids content of preg. soln	< 10 p.p.m.		20 - 1000 p.p.m.
Concentration/Purification	SX(Purlex) in pump mixers and settlers	Ion-exchange in fixed-bed columns	Continuous ion exchange (NIMCIX) + SX in Bufflex configuration.
Configuration (SX, IX)	4 stages extraction, 5 stages scrubbing, 4 stages stripping	2 stages on absorption, 1 elution	CIX - 1 absorption, 1 elution both continuous
Resin /Solvent	1 stage solvent regen. Tertiary amine solvent	Conventional strong-base resin	SX : 4 stages extr. 5 stages scrub. Conv. Strong-base resin tertiary amine solvent
Elution /Stripping	Controlled NH ₃ strip	Ammonium nitrate elution	10% H ₂ SO ₄ elution, Controlled pH strip with ammonia
Precipitation Type	Uranium precip. with NH ₃ gas in conv. stirred tanks	Intermediate impurity ppt with CaO followed by uranium ppt with ammonia	Uranium precipitation with NH ₃ gas in conv. stirred tanks
Drying/Calcination			
Product	Yellowcake in slurry transported by road tanker to central calcining plant (NUFCOR)		
Unusual Features	December 1978		

314-12	RIO TINTO - PALABORA	Transvaal (Pretoria)	SOUTH AFRICA
Owner/Operator	Rio Tinto Zinc Corp. + Newmont Mining + Industrial Develop. Corp./Palabora Mining Company.		
Stage of Exploitation	Routine production. Start up 1971.		
Grade (Range)	0.004% U ₃ O ₈ (40 p.p.m.) in original carbonatite ore.		
Production (Range)	Approximate, 100 t U/a.		
Deposit Ore Type	Carbonatite - The Phalaborwa Complex consists of pyroxenite, syenite, and a centrally located ore of transgressive carbonatite which is surrounded by a serpentine (olivine) - magnetite - apatite rock termed phoscorite.		
Uraniferous Minerals	Small concentrations of uranium and thorium occur in the phoscorite and carbonatite as uranothorianite.		
Other Minerals	Copper occurs principally as chalcocite, bornite, chalcocite, cubanite, and valierite also occur. Baddeleyite (zirconium oxide) is present in small amounts.		
Mining	Copper, apatite and vermiculite are mined on a large scale.		
Sorting/Blending	No sorting.		
Preconcentration	The flotation tailings from the copper concentrator, after desliming, is subjected to gravity concentration in Reichert cones to give a 40-50 fold concentration of heavy-minerals fraction, grav. conc. is fed to shaking tables where uranothorianite is separated from other heavy minerals.		
Crushing	-		
Grinding	-		
Degree of Comminution	Comminution of the run of the mine ore to about 50% minus 200-mesh is adequate to liberate the copper sulphides for flotation. No further grinding of stream from which uranothorianite conc. is produced		
Leach Type	Hot nitric acid leach in mechanical agitators.		
Oxidant/Temperature (Range)	No oxidant added 70°C		
Leach Pulp Density (Range)	-		
Remarks on Leaching	Dissolution of uranium and thorium occurs but not of zirconium. Fumes (oxides of nitrogen) from the leach are scrubbed with ammonium hydroxide.		
Solids/Liquid Separation	Tilting pan filters.		
Clarification	Thickener + centrifuge for clarification		
Solids content of preg. soln	< 20 p.p.m.		
Concentration/Purification	Solvent extraction in mixer-settlers.		
Configuration (SX, IX)	6 stage extraction, 4 stage scrub, 3 stage strip.		
Resin /Solvent	10% TBP solvent, thorium scrubbing with pure uranyl nitrate solution.		
Elution /Stripping	Warm water for stripping.		
Precipitation Type	Precipitation with ammonia in stirred tanks.		
Drying/Calcination	Skinner Roaster.		
Product	Calcined uranium concentrates in drums.		
Unusual Features	Lowest grade ore being treated for uranium recovery in the world. Only economic because of success preconcentration and because uranium is a minor by-product of copper production.		

4 0 1 - 1	S. U. B. - FORSTAU	Salzburg (Radstadt Area)	AUSTRIA
Owner/Operator	S.U.B.		
Stage of Exploitation	In the evaluation stage.		
Grade (Range)	0.05 - 0.10% U_3O_8 .		
Production (Range)	Not known yet what future production rate might be.		
Deposit Ore Type	Sandstone - Tabular orebody within a series of sericitic quartzites and sericite - chlorite schists. Uraniferous zone contains small individual lenticular orebodies.		
Uraniferous Minerals	Finely dispersed pitchblende and uraninite are concentrated in carbonaceous material.		
Other Minerals	Quartz, mica, feldspars.		
Mining	Mine development work in an advanced stage.		
Sorting/Blending			
Preconcentration			
Crushing			
Grinding			
Degree of Comminution			
Leach Type			
Oxidant/Temperature (Range)			
Leach Pulp Density (Range)			
Remarks on Leaching			
Solids/Liquid Separation			
Clarification			
Solids content of preg. soln			
Concentration/Purification			
Configuration (SX, IX)			
Resin /Solvent			
Elution /Stripping			
Precipitation Type			
Drying/Calcination			
Product			
Unusual Features	No details published on metallurgical response of ore as yet.		April 1978

4 0 2 - 1	R I B B - K V A N E F J E L D	Greenland (Ilulissat)	DENMARK
Owner/Operator	Geological Survey of Greenland and Danish Atomic Energy Commission		
Stage of Exploitation	Pilot plant evaluation. 200 tons ore produced from test shaft.		
Grade (Range)	0.05 - 0.10% U_3O_8		
Production (Range)	500 - 1000 t U/a		
Deposit Ore Type	An intrusion composed of an early augite syenite followed by strongly peralkaline, esgpeitic syenites the latter being accompanied by hydrothermal veins		
Uraniferous Minerals	Steenstrupine, Monazite, Thorite, Eudialyte, Pyrochlore, Rinkite.		
Other Minerals	Britholite, Epistilite, Lomonosovite, Lueshite, Murmanite, Chkalovite, Sorensenite, Tugtupite, Villiamite. The Lujavrite has plates of microcline in a matrix consisting of fine grained arfvedsonite. Sodolites etc. comprise minor components.		
Mining	Underground mining likely		
Sorting/Blending	Not investigated. Radiometric sorting a possibility		
Preconcentration	Flotation being investigated		
Crushing	Crushing will be required. Details not decided		
Grinding	Milling in presence of lixiviant under consideration.		
Degree of Comminution	Not decided		
Leach Type	Alkaline pressure leaching. Pipe Reactor technique under investigation		
Oxidant/Temperature (Range)	O_2 (under pressure) / 300°C 83 atmospheres		
Leach Pulp Density (Range)	50% solids		
Remarks on Leaching	The possible saturation of recycle solutions with fluorides could provide a problem		
Solids/Liquid Separation	Rotary drum vacuum filters being investigated		
Clarification	No clarification		
Solids content of preg. soln	20 - 1000 p.p.m.		
Concentration/Purification	Direct precipitation from alkaline leach liquors		
Configuration (SX, IX)	-		
Resin /Solvent	-		
Elution /Stripping	-		
Precipitation Type	H_2 ppt under pressure of 15 atmospheres and at 150°C. Catalysis by sintered UO_2 .		
Drying/Calcination	Not decided		
Product	UO_2 precipitate		
Unusual Features			June 1978.

404 - 1	SIMO - BESSINES	Limousin (Crouzille)	FRANCE
Owner/Operator	Cogema - PCUK - Caisse des depots et consignations/Societe Industrielle des Minerais de l'Ouest (SIMO)		
Stage of Exploitation	Routine Production. Start up 1958.		
Grade (Range)	0.12 - 0.24% U_3O_8 .		
Production (Range)	2000 - 2200 t/d. 1000 t U/a.		
Deposit Ore Type	Vein.		
Uraniferous Minerals	Pitchblende, coffinite, sobugelite, autunite, gummite, uranospathe, phosphuranylite and 8 uranotile.		
Other Minerals	Pyrite, chalcocopyrite, galena, hematite, melnikorite, enargite, calcite, clays.		
Mining	Underground and open pit.		
Sorting/Blending	No sorting.		
Preconcentration	No preconcentration.		
Crushing	Jaw for primary, gyratory crushers for secondary and tertiary.		
Grinding	First line has 2-stage open circuit with rod and ball mills. Second line has 2-stage (rod and ball) mills in closed circuit with classifiers.		
Degree of Comminution	95% minus 450 microns.		
Leach Type	Acid leach.		
Oxidant/Temperature (Range)	Sodium chlorate. 60°C.		
Leach Pulp Density (Range)	First line - 55% solids Second line - 55% solids for sands, 45% solids for fines.		
Remarks on Leaching	Two parallel leaching circuits. - leaching of sands in rotary drums. - leaching of fines in mechanical agitators.		
Solids/Liquid Separation	First line - horizontal belt filters for sands. Second line - 5 stages CCD in conventional thickeners for fines.		
Clarification	- rake classifiers for sands.		
Solids content of preg. soln	Downflow anthracite coal to give pregnant solution with < 20 p.p.m. solids.		
Concentration/Purification	First line - solvent extraction for pregnant solutions with high uranium content. Second line - Fixed bed ion exchange in Eluex/Buflex configuration (IX + SX) for low uranium content pregnant solutions.		
Configuration (SX, IX)	SX 4 - stage extraction, 3 stages stripping. (IX + SX) - conventional configuration.		
Resin/Solvent	Tertiary amine solvent. Conventional strong base resin.		
Elution/Stripping	Controlled pH ammonia strip with ammonium sulphate recycle. 10% H_2SO_4 elution.		
Precipitation Type	Ammonium hydroxide for precipitation in stirred tanks.		
Drying/Calcination	Rotary rolls heated with steam.		
Product	Dried yellowcake in drums.		
Unusual Features	Idea of doing sand/slime separation and treating each fraction separately in the optimum type of equipment. April 1978.		

404 - 2	SIMO - SAINT PRIEST	Lyonsais (Forez)	FRANCE
Owner/Operator	Compagnie Generale des Matieres Nucleaires (Cogema)/Societe Industrielle de Minerais de l'Ouest (SIMO)		
Stage of Exploitation	Start up 1960. Mill nearing end of life. Shutdown 1980.		
Grade (Range)	0.30 - 0.40% U_3O_8 .		
Production (Range)	500 t/d. 450 t U/a.		
Deposit Ore Type	Vein.		
Uraniferous Minerals	Pitchblende, coffinite, torbernite, uranophane, phosphuranylite.		
Other Minerals	Pyrite, chalcocopyrite, galena, hematite, quartz, fluorite.		
Mining	Underground and open pit mining.		
Sorting/Blending	None		
Preconcentration	None		
Crushing	Primary jaw crushers, secondary and tertiary gyratory crushers.		
Grinding	Two stage rod mills in closed circuit with classification.		
Degree of Comminution	95% minus 450 microns.		
Leach Type	Acid leach.		
Oxidant/Temperature (Range)	Sodium chlorate. 65°C.		
Leach Pulp Density (Range)	50% solids.		
Remarks on Leaching			
Solids/Liquid Separation	Line 1 - rotary drum vacuum filters. Line 2 - horizontal belt filters.		
Clarification	No clarification.		
Solids content of preg. soln	20 - 1000 p.p.m.		
Concentration/Purification	Intermediate impurity precipitation before uranium precipitation. Solvent Extraction		
Configuration (SX, IX)	SX has 4 stages extraction in mixer settlers. Pulse column for stripping (equivalent to 5 stages)		
Resin/Solvent	Tributyl phosphate (TBP)		
Elution/Stripping	Water		
Precipitation Type	Not applicable		
Drying/Calcination	Not applicable		
Product	Concentrated uranyl nitrate solution in vessels transported by rail.		
Unusual Features	Only uranium mill in the world to use TBP SX. October 1978.		

404 - 3	SINO - L'ECARPIERE	Poitou (Vendée)	FRANCE
Owner/Operator	Société Industrielle des Minéraux de l'Ouest (SINO) / Pechiney Ugine Kuhlman		
Stage of Exploitation	Routine production. Start up 1957.		
Grade (Range)	0.12 - 0.24% U_3O_8		
Production (Range)	1000 - 1200 t/d 800 t U/a.		
Deposit Ore Type	Vein.		
Uraniferous Minerals	Pitchblende, coffinite, johannite.		
Other Minerals	Pyrite, hematite, fluorite, quartz.		
Mining	Open pit and underground mining.		
Sorting/Blending	No sorting		
Preconcentration	No preconcentration.		
Crushing	Jaw crushers for primary, gyratory crushers for secondary and tertiary.		
Grinding	Two stage open circuit with rod and ball mills.		
Degree of Comminution	95% minus 450 microns.		
Leach Type	Acid leach.		
Oxidant/Temperature (Range)	Sodium chlorate. 60°C.		
Leach Pulp Density (Range)	60% solids		
Remarks on Leaching	High terminal acid of 40g/l		
Solids/Liquid Separation	Single stage rake classifiers to effect prior separation at 150 microns.		
Clarification	3 stage rake classifiers for sands washing. 6 stage CCD in conventional thickeners for fines.		
Solids content of preg. soln	Downflow columns with anthracite coal. < 20 p.p.m.		
Concentration/Purification	Fixed bed ion exchange in Eluex/Bufflex configuration (IX + SX)		
Configuration (SX, IX)	IX - single stage absorption, single stage elution. SX - 4 stage extraction, 3 stage stripping.		
Resin/Solvent	Conventional strong base resin. Tertiary amine solvent.		
Elution/Stripping	10% H_2SO_4 elution. Controlled pH ammonia stripping.		
Precipitation Type	Ammonium hydroxide for uranium precipitation in stirred tanks.		
Drying/Calcination	Belt type dryer.		
Product	Dried yellowcake in drums.		
Unusual Features			

April 1978.

404 - 4	PCUK - ST PIERRE LE CANTAL	Massif Central	FRANCE
Owner/Operator	Pechiney Ugine Kuhlmann - Compagnie Française des Pétroles (Minatome) Pechiney Ugine Kuhlmann		
Stage of Exploitation	Routine production, start up 1977.		
Grade (Range)	0.12 - 0.24% U_3O_8 .		
Production (Range)	500 t/d. 100 t U/a.		
Deposit Ore Type	Sandstone - arkosic rocks.		
Uraniferous Minerals	Francevillite, uranocircite.		
Other Minerals	Clays.		
Mining	Open pit mining.		
Sorting/Blending	No sorting. No blending.		
Preconcentration	No preconcentration.		
Crushing	No crushing		
Grinding	No milling.		
Degree of Comminution	Run of mine ore is very fine - screening and cycloning used to obtain size cut near 80 μm		
Leach Type	Acid leach.		
Oxidant/Temperature (Range)	No oxidant added. 60°C.		
Leach Pulp Density (Range)	30 - 40% (low because of clay content).		
Remarks on Leaching	Line 1 - Vat leaching for +80 μm Line 2 - Leaching in mechanical agitators for -80 μm		
Solids/Liquid Separation	None (rheological properties make settling and filtration difficult.)		
Clarification	None.		
Solids content of preg. soln	Dilute pulp of about 1% solids to CIX.		
Concentration/Purification	Pechiney Ugine Kuhlman CIX (slime feed).		
Configuration (SX, IX)	Single column for absorption, single column for elution.		
Resin/Solvent	Big bead strong-base resin.		
Elution/Stripping	Sodium chloride elution.		
Precipitation Type	Magnesia for uranium precipitation from concentrated eluate solution.		
Drying/Calcination	Belt type dryer.		
Product	Dried yellowcake in drums.		
Unusual Features	Slime fed upflow CIX column		

April 1978

4 0 4 - 5	C.F.M.U. - L O Z E R E	Languedoc (Hérault)	FRANCE
Owner/Operator	C.F.M.U.		
Stage of Exploitation	Previously heap leaching operation for 10 years. New conventional plant started up 1976.		
Grade (Range)	0.12 - 0.24% U_3O_8 .		
Production (Range)	600 t/d. 180 t U/a.		
Deposit Ore Type	Sandstone		
Uraniferous Minerals	Pitchblende, coffinite, uraniferous organic material, uranium molybdate, uranophane, saleite, novacekite, billietite, zippeite, curite, zinnerite.		
Other Minerals	Pyrite, uranothite, wulfenite, erythrite, chalcocite, sphalerite, dolomite, pelites and bituminous shale.		
Mining	Open pit and underground mining.		
Sorting/Blending	No sorting. No blending.		
Preconcentration	No preconcentration.		
Crushing	Jaw for primary and gyratory for secondary and tertiary crushing.		
Grinding	Two stage rod and ball mills in closed circuit with classification		
Degree of Comminution	95% minus 500 microns.		
Leach Type	Acid leach.		
Oxidant/Temperature (Range)	Sodium chlorate. 50°C.		
Leach Pulp Density (Range)	50% solids.		
Remarks on Leaching	Horizontal belt filter for dewatering of milled pulp prior to leaching. Repulp done in first acid leach vessel.		
Solids/Liquid Separation	Horizontal belt filter (85m ² area).		
Clarification	Downflow column with anthracite coal.		
Solids content of preg. soln	<20 p.p.m.		
Concentration/Purification	Solvent extraction in Krebs mixer-settlers.		
Configuration (SX, IX)	4 stage extraction, 5 stage stripping.		
Resin /Solvent	Tertiary amine solvent.		
Elution /Stripping	Sodium chloride stripping.		
Precipitation Type	Magnesia used for uranium precipitation in stirred tanks.		
Drying/Calcination	Belt type dryer.		
Product	Dried yellowcake in drums.		
Unusual Features	Of special interest is the successful use of horizontal belt filters for dewatering and solids/liquids separation.		April 1978

4 0 4 - 6	COGEMA - L O D È V E	Languedoc (Hérault)	FRANCE
Owner/Operator	Compagnie Générale des Matières Nucieaires (COGEMA)		
Stage of Exploitation	Construction/Commissioning. Start-up scheduled for 1980		
Grade (Range)	0.40 - 0.50% U_3O_8		
Production (Range)	1400 - 1500 t/d. 900 - 1000 t U/a		
Deposit Ore Type	Sandstone - uranium occurs in pelites, sandstone and bituminous shale		
Uraniferous Minerals	Pitchblende, coffinite, uraniferous organic material, uranium molybdate, saleite, novacekite, billietite, uranophane, zippeite, curite, zinnerite		
Other Minerals	Pyrite, uranothite, wulfenite, erythrite, chalcocite, sphalerite, dolomite.		
Mining	Underground		
Sorting/Blending	No sorting		
Preconcentration	No preconcentration		
Crushing	Jaw for primary, gyratory crushers for secondary.		
Grinding	Two stage grind in closed circuit with classification using rod and ball mills		
Degree of Comminution	95% minus 200 microns		
Leach Type	Alkaline pressure leach in autoclaves (pressure 6 bars)		
Oxidant/Temperature (Range)	Oxygen + ferric carbonate /140°C		
Leach Pulp Density (Range)	50% solids		
Remarks on Leaching	Differential leach. First stage is a dilute alkaline pressure pre-leach principally designed to convert sulphur in ore to sulphates, second stage is a more-concentrated alkaline pressure leach in which most of the uranium is extracted without solubilizing the organic components in the ore.		
Solids/Liquid Separation	Two stages of filtration using horizontal belt vacuum filters		
Clarification	No clarification		
Solids content of preg. soln	20 - 1000 p.p.m.		
Concentration/Purification	Intermediate impurity precipitation with calcium hydroxide directly from alkaline leach liquors followed by uranium precipitation with hydrogen peroxide in conventional stirred tanks		
Configuration (SX, IX)	-		
Resin /Solvent	-		
Elution /Stripping	-		
Precipitation Type	See above		
Drying/Calcination	Not decided yet		
Product	Uranium peroxide		
Unusual Features			July 1978

4 0 0 - 1	ENU - NISA	Alto Alentejo (Nisa Area)	PORTUGAL
Owner/Operator	Empresa Nacional de Uranio (ENU)		
Stage of Exploitation	Under Investigation. Not yet in production.		
Grade (Range)	0.10 - 0.15% U_3O_8		
Production (Range)	Not applicable.		
Deposit Ore Type	Vein - disseminated.		
Uraniferous Minerals	Salesite, autunite, phosphoranylite, pitchblende (rare).		
Other Minerals	Quartz, sericite, biotite, plagioclase, iron oxide and hydroxide, graphite, zircon, tourmaline.		
Mining			
Sorting/Blending			
Preconcentration			
Crushing			
Grinding			
Degree of Comminution			
Leach Type			
Oxidant/Temperature (Range)			
Leach Pulp Density (Range)			
Remarks on Leaching			
Solids/Liquid Separation			
Clarification			
Solids content of preg. soln			
Concentration/Purification			
Configuration (SX, IX)			
Resin /Solvent			
Elution /Stripping			
Precipitation Type			
Drying/Calcination			
Product			
Unusual Features	No details published on metallurgical response of ore as yet.		

April 1976

4 0 0 - 1	ENUSA - FE	Salamanca (Saelices al chico)	SPAIN
Owner/Operator	Empresa Nacional del Uranio S.A. (ENUSA)		
Stage of Exploitation	Heap leach in operation. Conventional plant planned for start up in 1980.		
Grade (Range)	0.05 - 0.10% U_3O_8 .		
Production (Range)	Heap leach 120 t U/a. New plant 3000 t/d and 400 - 500 t U/a.		
Deposit Ore Type	Vein - Disseminated deposit in shales which have some organic matter, have been little affected regional metamorphism, but have been altered by contact metamorphism to hornfels.		
Uraniferous Minerals	Pitchblende, coffinite, uranophane, kasolite, gummite, autunite.		
Other Minerals	Pyrite, hematite, illite.		
Mining	Open pit mining. With radiometric control at the pit. Front end loaders and bucket-wheel excavators used for dump reclamation.		
Sorting/Blending	Three types of product : mineral, marginal ore and waste.		
Preconcentration	No preconcentration. For new plant ore will be roasted at 350 - 400°C prior to acid leach.		
	HEAP LEACH OPERATION	NEW CONVENTIONAL PLANT	
Crushing	Primary crushing with impact crusher only and no milling.	Primary jaw crushers.	
Grinding		Single stage Aerofall mill in closed circuit with classification.	
Degree of Comminution	All minus 50 mm.	80% minus 48- mesh(50% minus 65 mesh.)	
Leach Type	Acid heap leaching.	Acid leaching in mechanical agitators.	
Oxidant/Temperature (Range)	No oxidant addition. Ambient temp.	$MnO_2 + Fe_2(SO_4)_3$.	
Leach Pulp Density (Range)	Not applicable.	50 - 55.	
Remarks on Leaching			
Solids/Liquid Separation	None	5 stage CCD in conventional thickeners	
Clarification	None	Downflow sand clarifiers (columns)	
Solids content of preg. soln	< 100 p.p.m.	< 20 p.p.m.	
Concentration/Purification	Solvent extraction	Solvent extraction	
Configuration (SX, IX)			
Resin /Solvent	Tertiary amine solvent.	Tertiary amine solvent.	
Elution /Stripping	$NaCl + H_2SO_4$	Controlled pH ammonia strip.	
Precipitation Type	NH_3 gas + air in stirred tanks	NH_3 gas + air in stirred tanks	
Drying/Calcination	Gas fired oven	Multiple hearth	
Product	Calcined uranium oxide concentrates in drums	Calcined uranium oxide concentrates in drums	
Unusual Features		Low temperature roasting to open the uraniferous limonites.	
October 1976.			

4 0 0 - 2	JEN - EL PEDREGAL	Badajoz (La Haba)	SPAIN
Owner/Operator	Junta de Energía Nuclear (JEN)		
Stage of Exploitation	In operation on an experimental basis.		
Grade (Range)	0.05 - 0.15% U_3O_8		
Production (Range)	< 100 t/a		
Deposit Ore Type	Vein - disseminated deposit in shales, which have some organic matter. Has been little affected by regional metamorphism, but has been altered by contact metamorphism.		
Uraniferous Minerals	Pitchblende, gummite, autunite, salesite and subugolite.		
Other Minerals	Pyrite, chalcocopyrite, quartz, hematite and illite.		
Mining	Open pit mining with radiometric control at the pit. Front end loader used for dump reclamation.		
Sorting/Blending	Four types of product: rich mineral (to be sent to Andujar), average mineral, marginal ore, and waste.		
Preconcentration	No preconcentration.		
Crushing	Primary jaw crushers for "mineral" fraction only, screening and wet classification into two fractions.		
Grinding	Coarse (+ 20- mesh) Fines		
Degree of Comminution	minus 50 mm 90% minus 35 - mesh (50% minus 100 - mesh)		
Leach Type	Acid heap leaching. Acid leach in mechanical agitators.		
Oxidant/Temperature (Range)	No oxidant / ambient temperature. No oxidant / ambient temperature.		
Leach Pulp Density (Range)	Not applicable 45 - 50% solids		
Remarks on Leaching			
Solids/Liquid Separation	None. Sand-slime separation, with 3 stages washing in hydrocyclones		
Clarification	None. None.		
Solids content of preg. soln			
Concentration/Purification	Solvent extraction of the blend of marginal ore and coarse pregnant liquors and the eluates from RIP-IX. Screen-mix RIP.		
Configuration (SX, IX)			
Resin /Solvent	Tertiary amine solvent. Large size strong-base IX resin.		
Elution /Stripping	NaCl + H_2SO_4 10% sulphuric acid.		
Precipitation Type	NH_3 gas + air in stirred tanks		
Drying/Calcination	Multiple hearth.		
Product	Calcined uranium - oxide concentrates in drums.		
Unusual Features	Experimental flowsheet includes heap leaching of coarse material, agitation leach of the fines and RIP. October 1978.		

4 0 0 - 3	JEN - LOS BATIÑOS JEN - ANDUJAR	Caceres (Albala)	SPAIN
Owner/Operator	Junta de Energía Nuclear (JEN)		
Stage of Exploitation	Production.		
Grade (Range)	0.15 - 0.20% U_3O_8		
Production (Range)	< 100 t/a.		
Deposit Ore Type	Vein		
Uraniferous Minerals	Pitchblende, gummite, autunite, salesite.		
Other Minerals	Pyrite, quartz, hematite, apatite.		
Mining	Underground selective mining with ore sent to Andujar mill		
Sorting/Blending	Blending with ores from other sources.		
Preconcentration	No - sometimes low temperature (300 - 400°C) roasting.		
Crushing	Two stage (jaw and cone) crushing.		
Grinding	Rod and ball milling in closed circuit with hydraulic classification.		
Degree of Comminution	80% minus 26 mesh (50% minus 100 -mesh)		
Leach Type	Acid leaching in mechanical agitators.		
Oxidant/Temperature (Range)	No oxidant / 40 - 50.		
Leach Pulp Density (Range)	50 - 60% solids.		
Remarks on Leaching			
Solids/Liquid Separation	4 stages, sands in cyclones and fines in CCD in conventional thickeners.		
Clarification	Downflow sand clarifiers.		
Solids content of preg. soln	< 20 p.p.m.		
Concentration/Purification	Solvent extraction.		
Configuration (SX, IX)			
Resin /Solvent	Tertiary amine solvent.		
Elution /Stripping	NaCl and H_2SO_4 .		
Precipitation Type	Ammonia in one stage batch operation.		
Drying/Calcination	Electrically heated oven.		
Product	Dried uranium-oxide concentrate in drums.		
Unusual Features			
	October 1978.		

4 0 0 - 4	ENUSA - MAZARETE	Guadalajara (Molina de Aragón)	SPAIN
Owner/Operator	Junta de Energía Nuclear / ENUSA		
Stage of Exploitation	Evaluation		
Grade (Range)	0.05 - 0.10% U_3O_8		
Production (Range)	Not in production yet. (100 - 160 t U/a)		
Deposit Ore Type	Sandstone		
Uraniferous Minerals	Pitchblende, autunite, torbernite, carnotite,		
Other Minerals	Pyrite, chalcocopyrite, quartz, hematite, dolomite.		
Mining	Open pit.		
Sorting/Blending	No		
Preconcentration	No		
Crushing	Two stage (jaw and cone) crushing.		
Grinding	Ball milling in closed circuit.		
Degree of Comminution	80% minus -48 mesh. (50% minus 65 mesh).		
Leach Type	Acid leaching in mechanical agitators.		
Oxidant/Temperature (Range)	Manganese dioxide / ambient temperature.		
Leach Pulp Density (Range)	50 - 55 % solids		
Remarks on Leaching			
Solids/Liquid Separation	Horizontal belt filters.		
Clarification	Downflow sand clarifiers.		
Solids content of preg. soln	< 20 p.p.m.		
Concentration/Purification	Solvent extraction.		
Configuration (SX, IX)			
Resin /Solvent	Tertiary amine solvent.		
Elution /Stripping	Controlled pH ammonia strip.		
Precipitation Type	NH_3 gas and air in stirred tanks.		
Drying/Calcination	Multiple hearth.		
Product	Calcined uranium oxide concentrates in drums.		
Unusual Features			

October 1978.

4 0 0 - 5	FESA - HUELVA	Huelva	SPAIN
Owner/Operator	Fosforico Español Incorporated (FESA)		
Stage of Exploitation	Environmental impact.		
Grade (Range)	120 - 140 g U_3O_8/m^3		
Production (Range)	Not in production yet. (100 t U/a)		
Deposit Ore Type	From international phosphatic rock (mainly from Morocco and W. Sahara).		
Uraniferous Minerals			
Other Minerals			
Mining			
Sorting/Blending			
Preconcentration			
Crushing			
Grinding	Dry ball milling.		
Degree of Comminution			
Leach Type	Acid leaching in mechanical agitators.		
Oxidant/Temperature (Range)	/ 55 - 65°C		
Leach Pulp Density (Range)			
Remarks on Leaching	Preyon process.		
Solids/Liquid Separation	Tilting pan filter.		
Clarification	Tank ageing, decantation in thickeners and downflow columns		
Solids content of preg. soln	Low content in solids (< 50 p.p.m.) in the final pregnant liquors.		
Concentration/Purification	Solvent extraction.		
Configuration (SX, IX)	Two cycles, the first for recovery and the second for purification.		
Resin /Solvent	TDPO + DBHPA		
Elution /Stripping	Reduction media (1st cycle) and carbonate (2nd cycle)		
Precipitation Type	Crystallization of AUTC.		
Drying/Calcination	Single hearth.		
Product	Calcined uranium oxide in drums.		
Unusual Features	Recovering uranium from phosphoric acid.		

October 1978.

4 0 9 - 6	ENUSA - CARIDAD	Salamanca (Villavieja de Yebes)	SPAIN
Owner/Operator	Empresa Nacional del Uranio S.A. (ENUSA)		
Stage of Exploitation	Environmental impact.		
Grade (Range)	0.06 - 0.15% U_3O_8		
Production (Range)	Not in production yet. (~ 40 t U/a)		
Deposit Ore Type	Vein - Disseminated deposit in shale, ore is rich in organic matter, has been little affected by original metamorphism, but has been altered by contact metamorphism to hornfels.		
Uraniferous Minerals	Pitchblende, gumite, torbernite, salesite and uranopilite		
Other Minerals	Pyrite, quartz, quartzite-limonite, sericite.		
Mining	Open pit mining on a temporary contract basis once a year.		
Sorting/Blending	No		
Preconcentration	No		
Crushing	Primary crushing with mobile plant on a temporary basis once a year.		
Grinding	No		
Degree of Comminution	All minus 50 mm.		
Leach Type	Acid heap leaching.		
Oxidant/Temperature (Range)	No oxidant addition / ambient temperature.		
Leach Pulp Density (Range)	Not applicable.		
Remarks on Leaching			
Solids/Liquid Separation	None		
Clarification	None		
Solids content of preg. soln	<100 p.p.m.		
Concentration/Purification	Solvent extraction		
Configuration (SX, IX)	(modular mixer-settler plant).		
Resin /Solvent	Tertiary amine solvent.		
Elution /Stripping	NaCl and H_2SO_4 .		
Precipitation Type	Ammonium hydroxide.		
Drying/Calcination	No.		
Product	ADU slurry in road tankers.		
Unusual Features	Modular plant suited to small deposits.		

October 1978.

4 1 0 - 1	LKAB - RANSTAD	Västergötland (Bjällinge)	SWEDEN
Owner/Operator	Luossavaara - Kirunaavaara AB, AB Atomenergi and the Swedish Power Board		
Stage of Exploitation	Evaluation. Large pilot plant in intermittent operation since 1965.		
Grade (Range)	~ 0.05% U_3O_8		
Production (Range)	1000 - 2000 t U/a		
Deposit Ore Type	Bituminous shales		
Uraniferous Minerals	Uranium associated with organic material		
Other Minerals	Pyrite, quartz, illite, and feldspars		
Mining	Open pit proposed		
Sorting/Blending	No sorting		
Preconcentration	Dense media rejection of limestone		
Crushing	Primary crushing underground (gratory)		
Grinding	Hammer mill		
Degree of Comminution	100% minus 2mm		
Leach Type	Strong acid vat leaching		
Oxidant/Temperature (Range)	No oxidant / 60°C		
Leach Pulp Density (Range)	14% moisture (by weight)		
Remarks on Leaching	Contact with H_2SO_4 in curing drums followed by countercurrent percolation in vats is latest process under investigation.		
Solids/Liquid Separation	Vat bottoms contain gravel to provide filtration and clarification.		
Clarification			
Solids content of preg. soln	< 20 p.p.m.		
Concentration/Purification	Solvent extraction in mixer settlers		
Configuration (SX, IX)	Not published		
Resin /Solvent	Tertiary amine solvent		
Elution /Stripping	Sodium carbonate/ammonium carbonate stripping		
Precipitation Type	Uranium precipitation from strip solutions with sodium hydroxide		
Drying/Calcination	Not published		
Product	Yellowcake (sodium diuranate)		
Unusual Features	Deposit located in a densely populated and fairly extensively cultivated district. Very stringent environmental constraints imposed by authorities.		

July 1978

413-1	BORIS KIDRIC - ZIROVSKI VRH	Northwest Yugoslavia	YUGOSLAVIA
Owner/Operator			
Stage of Exploitation			
Grade (Range)			
Production (Range)			
Deposit Ore Type	Sandstone		
Uraniferous Minerals	Variety of uranium silicates		
Other Minerals			
Mining			
Sorting/Blending			
Preconcentration			
Crushing			
Grinding			
Degree of Comminution			
Leach Type			
Oxidant/Temperature (Range)			
Leach Pulp Density (Range)			
Remarks on Leaching			
Solids/Liquid Separation			
Clarification			
Solids content of preg. soln			
Concentration/Purification			
Configuration (SX, IX)			
Resin /Solvent			
Elution /Stripping			
Precipitation Type			
Drying/Calcination			
Product			
Unusual Features	Very little published on response of ore to treatment methods		
	April 1978		

501-1	RANGER - JABIRU	Northern Territory (Alligator Rivers)	AUSTRALIA
Owner/Operator	Geopeko + Electrolytic Zinc + State/Ranger Uranium Mines Ltd.,		
Stage of Exploitation	Environmental impact.		
Grade (Range)	0.20 - 0.30% U_3O_8		
Production (Range)	3000 - 4000 t/d. 2000 - 3000 t U/a.		
Deposit Ore Type	Vein - uranium mineralization occurs in sediments of the Pine Creek Geosyncline within the Cahill Formation.		
Uraniferous Minerals	Pitchblende, uraninite(major), uranium silicates, pseudo brannerite (minor)		
Other Minerals	Quartz, chlorite, muscovite, kaolinite.		
Mining	Open pit.		
Sorting/Blending	Blending from stockpiles		
Preconcentration	None		
Crushing	Gyratory crushers for primary, cones for secondary and tertiary.		
Grinding	Two-stage - open circuit rod mills followed by ball mills in closed circuit with classifiers.		
Degree of Comminution	50% minus 75 μ m.		
Leach Type	Acid leach.		
Oxidant/Temperature (Range)	Manganese dioxide/ambient		
Leach Pulp Density (Range)	55 - 60% solids		
Remarks on Leaching			
Solids/Liquid Separation	Five stage CCD in conventional thickeners.		
Clarification	Pressure sand filters.		
Solids content of preg. soln	< 20 p.p.m.		
Concentration/Purification	Solvent extraction in mixer-settlers		
Configuration (SX, IX)	4 stages extraction, 1 stage scrub, 4 stages strip.		
Resin /Solvent	Tertiary amine.		
Elution /Stripping	Controlled pH ammonia strip.		
Precipitation Type	Ammonia gas for uranium precipitation from strip solution, in stirred tanks.		
Drying/Calcination	Multiple hearth.		
Product	Uranium oxide concentrates in drums.		
Unusual Features			
	April 1978.		

501-2	NORANDA - KOONGARRA	Northern Territory (Alligator Rivers)	AUSTRALIA
Owner/Operator	Noranda Australia Ltd.		
Stage of Exploitation	Evaluation.		
Grade (Range)	0.40 - 0.50% U_3O_8		
Production (Range)	Not published.		
Deposit Ore Type	Vein - uranium mineralization occurs in sediments of the Pine Creek Geosyncline within the Cahill Formation.		
Uraniferous Minerals	Pitchblende, uraninite (major) Uranium phosphates and silicates (minor)		
Other Minerals	Quartz, chlorite, mica.		
Mining	Open pit (minor underground mining)		
Sorting/Blending			
Preconcentration			
Crushing			
Grinding			
Degree of Comminution			
Leach Type	Acid leach proposed.		
Oxidant/Temperature (Range)			
Leach Pulp Density (Range)			
Remarks on Leaching			
Solids/Liquid Separation	CCD in conventional thickeners proposed.		
Clarification			
Solids content of preg. soln			
Concentration/Purification	Solvent extraction proposed.		
Configuration (SX, IX)			
Resin /Solvent			
Elution /Stripping			
Precipitation Type			
Drying/Calcination			
Product			
Unusual Features	Note - No detailed information published on flowsheet but it has been reported that the treatment process proposed is similar to that proposed for the Jabiru and Jabiluka ores.		

April 1978.

501-3	PANCONTINENTAL - JABILUKA	Northern Territory (Alligator Rivers)	AUSTRALIA
Owner/Operator	Pancontinental Mining + Getty Oil Development		
Stage of Exploitation	Environmental impact.		
Grade (Range)	0.30 - 0.40% U_3O_8		
Production (Range)	3000 - 4000 t/d. 3000 - 4000 t U/a.		
Deposit Ore Type	Vein - uranium mineralization occurs in sediments of the Pine Creek Geosyncline within the Cahill Formation.		
Uraniferous Minerals	Pitchblende, uraninite.		
Other Minerals	Quartz, chlorite, graphite, mica. Some ore contains gold.		
Mining	Open pit.		
Sorting/Blending	Stockpiles at mine. Blending before primary crusher.		
Preconcentration	None		
Crushing	Primary crushing at mine (gyratory). Secondary and tertiary cone crushers at mill.		
Grinding	Two stage circuit. Open circuit rod mills followed by ball mills in closed circuit with classifiers.		
Degree of Comminution	50% minus 75µm.		
Leach Type	Acid leach.		
Oxidant/Temperature (Range)	Manganese dioxide/ 35 - 45°C.		
Leach Pulp Density (Range)	55 - 60% solids		
Remarks on Leaching	Some ore contains gold which will be recovered by cyanidation before uranium recovery.		
Solids/Liquid Separation	CCD in conventional thickeners (no of stages not published)		
Clarification	Pressure sand clarifiers		
Solids content of preg. soln	< 20 p.p.m.		
Concentration/Purification	Solvent extraction in mixer settlers.		
Configuration (SX, IX)	4 stages extraction, 4 stages stripping (scrub not known).		
Resin /Solvent	Tertiary amine		
Elution /Stripping	Controlled pH ammonia strip.		
Precipitation Type	Ammonia gas for uranium precipitation from strip solution, in stirred tanks.		
Drying/Calcination	Multiple hearth		
Product	Dried yellowcake in drums.		
Unusual Features	The two Jabiluka orebodies together represent the largest known uranium deposit in the world.		

April 1978

5.01 - 4	QUEENSLAND MINES - NABARLEK	Northern Territory (Alligators Rivers)	AUSTRALIA
Owner/Operator	Queensland Mines Ltd.		
Stage of Exploitation	Environmental impact.		
Grade (Range)	2.00 - 3.00% U_3O_8		
Production (Range)	< 500 t/d 1000 - 2000 t U/a.		
Deposit Ore Type	Vein - uranium mineralization occurs in sediments of the Pine Creek Gossyncline within the Cahill Formation		
Uraniferous Minerals	Pitchblende, uraninite (major) Skudowskite, coffinite, kasolite, zircon, pseudo brannerite.		
Other Minerals	Chlorite, hematite, sericite, kaolinite, quartz, muscovite.		
Mining	Open pit.		
Sorting/Blending	Stockpiles according to grade. Blending to produce uniform feed to plant.		
Preconcentration	Gravity concentration considered but not likely to be applied.		
Crushing	Jaw for primary. Cone for secondary crushing.		
Grinding	Single stage ball mill in closed circuit with classification.		
Degree of Comminution	50% minus 75 μ m		
Leach Type	Acid leach in mechanical agitators.		
Oxidant/Temperature (Range)	Manganese dioxide/ 35 - 45°C.		
Leach Pulp Density (Range)	55% solids.		
Remarks on Leaching			
Solids/Liquid Separation	8 stages CCD in conventional thickeners.		
Clarification	Deep bed sand filters.		
Solids content of preg. soln	< 20 p.p.m.		
Concentration/Purification	Solvent Extraction.		
Configuration (SX, IX)	4 extraction stages, 4 stripping stages, 1 scrub stage, 1 acidification stage.		
Resin /Solvent	Tertiary amine		
Elution /Stripping	Controlled pH ammonia strip.		
Precipitation Type	Ammonia gas for uranium precipitation from strip solution, in stirred tanks.		
Drying/Calcination	Multiple hearth calciner.		
Product	Uranium oxide concentrates in drums.		
Unusual Features	One of the highest grade uranium deposits in the world. Special problem with radioactivity during mining. Uranium concentration of leach liquors about 10 times higher than conventional practice thus pre-acidification of amine required in SX. April 1978.		

5.01 - 5	RIO TINTO - MARY KATHLEEN	Queensland (Mc Isa - Cloncurry)	AUSTRALIA
Owner/Operator	Consolidated Rio Tinto Ltd. + State / Mary Kathleen Uranium Ltd.		
Stage of Exploitation	Production.		
Grade (Range)	0.10 - 0.15% U_3O_8 .		
Production (Range)	1000 - 2000 t/d 500 - 1000 t U/a.		
Deposit Ore Type	Metasomatic - uranium mineralization occurs in host rocks that belong to the Corella Formation.		
Uraniferous Minerals	Uraninite.		
Other Minerals	Allanite, stilwellite, garnet, apatite, amphiboles, feldspar.		
Mining	Open pit.		
Sorting/Blending	Radiometric discriminator at mine for stockpiles. Radiometric sorting of feed to mill.		
Preconcentration	None (other than radiometric sorting).		
Crushing	Jaw for primary, standard cone for secondary, shorthead cone for tertiary.		
Grinding	Two stage circuit with classification. Rod mill plus 2 ball mills.		
Degree of Comminution	80% minus 200 mesh.		
Leach Type	Acid leach in mechanical agitators.		
Oxidant/Temperature (Range)	Manganese dioxide/ ambient		
Leach Pulp Density (Range)	55 - 60% solids		
Remarks on Leaching	4 - stage leach with pH controlled at 2.5, 2.0, and 1.0. Acid diluted and cooled before addition.		
Solids/Liquid Separation	4 stage CCD in conventional thickeners.		
Clarification	Downflow sand clarifiers.		
Solids content of preg. soln	< 20 p.p.m.		
Concentration/Purification	Solvent extraction in mixer settlers		
Configuration (SX, IX)	4 stage extraction, 1 scrub, 3 stripping		
Resin /Solvent	Tertiary amine.		
Elution /Stripping	Controlled pH ammonia strip		
Precipitation Type	Ammonia gas for uranium precipitation from strip solution in stirred tanks.		
Drying/Calcination	Multiple hearth		
Product	Uranium oxide concentrates in drums.		
Unusual Features			

April 1978.

501-6	WESTERN MINING - YEELEIRRIE	Western Australia (Yilgarn Block)	AUSTRALIA
Owner/Operator	Western Mining Corporation Ltd.		
Stage of Exploitation	Evaluation		
Grade (Range)	0.15 - 0.20% U_3O_8		
Production (Range)	No final decision on final production rate as yet		
Deposit Ore Type	Calcrete - uranium occurs in sediments comprising sand, clay and calcite in the Yilgarn Block		
Uraniferous Minerals	Carnotite		
Other Minerals	Calcite, dolomite, quartz, chert, clay minerals		
Mining	Open pit. Deposit is extremely shallow and mining will probably be with scrapers and bulldozer-strippers		
Sorting/Blending	Trucks from mine monitored radiometrically to allow graded stockpiles to be prepared. Blending to produce uniform feed to plant.		
Preconcentration	None		
Crushing	The ore will probably be crushed in impact crushers to about minus 20 mm then dried and roasted in rotary kilns. The roasted ore will be quenched in a spiral classifier and the underflow fed to the grinding circuit, which will probably consist of rod and ball mills in series.		
Grinding			
Degree of Comminution	80% minus 48 mesh.		
Leach Type	Alkaline carbonate		
Oxidant/Temperature (Range)			
Leach Pulp Density (Range)			
Remarks on Leaching	Details of the leaching conditions and equipment to be used in the full-scale plant have not been published but it has been reported that 95% uranium dissolution can be achieved by alkaline leaching at 125 degrees centigrade in outcrops.		
Solids/Liquid Separation	Clays present in ore result in poor solids/liquid separation.		
Clarification			
Solids content of preg. soln			
Concentration/Purification	No information published		
Configuration (SX, IX)	" " "		
Resin /Solvent	" " "		
Elution /Stripping	" " "		
Precipitation Type	" " "		
Drying/Calcination	" " "		
Product	" " "		
Unusual Features			

April 1978

501-7	WESTERN NUCLEAR-BEVERLEY	South Australia (Frome Embayment)	AUSTRALIA
Owner/Operator	Oilmin + Transoil + Patromin + Western Nuclear/Western Nuclear		
Stage of Exploitation	Evaluation		
Grade (Range)	0.20 - 0.30% U_3O_8		
Production (Range)	No published information on production plans		
Deposit Ore Type	Sandstone		
Uraniferous Minerals	Uraninite		
Other Minerals	Quartz, clay minerals		
Mining	Open pit proposed. Solution mining under consideration.		
Sorting/Blending			
Preconcentration			
Crushing			
Grinding			
Degree of Comminution			
Leach Type	Acid leach proposed		
Oxidant/Temperature (Range)			
Leach Pulp Density (Range)			
Remarks on Leaching			
Solids/Liquid Separation			
Clarification			
Solids content of preg. soln			
Concentration/Purification			
Configuration (SX, IX)			
Resin /Solvent			
Elution /Stripping			
Precipitation Type			
Drying/Calcination			
Product			
Unusual Features	No detailed information published on metallurgical response of ore, as yet.		

April 1978

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LIST OF CONSULTANTS

Meetings

- | | | |
|---|---|--|
| H.E. JAMES (<i>Chairman</i>) | January 1978
July 1978
October 1978 | Extraction Metallurgy Division
Atomic Energy Board,
Private Mail Bag X 2017,
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| H.W. FANDER | January 1978 | Central Mineralogical Services,
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| D. KURAT | January 1978 | Mineralogy and Petrology
Department,
Natural History Museum,
Vienna, Austria |
| D.C. SEIDEL | January 1978 | US Bureau of Mines,
Metallurgical Research Centre
Salt Lake City, Utah,
United States of America |
| H. SIMONSEN | July 1978 | Extraction Metallurgy Division
Atomic Energy Board,
Private Mail Bag X 2017,
Randburg 2125,
South Africa |
| F.R. HARTLEY
(<i>Scientific Secretary</i>) | 1978 | Division of Nuclear Power
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